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ERRATA VOL. VII.

Page 90, foot-note, *for* "20 million tons" *read* "20 million horse-power."

Page 246, line 23, *for* "fire-damp" *read* "fire-dam."

Page 329, line 22, *for* "professors" *read* "professor."

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PARTING GOLD AND SILVER IN THE UNITED STATES ASSAY OFFICE, NEW YORK CITY.

BY T. EGLESTON, PH.D.

THE Mints of the United States are situated in the following cities: Philadelphia, San Francisco, Carson City, Denver, and New Orleans. The Assay Offices are in New York, St. Louis, Helena, Boisé, and Charlotte. The Mints coin and, most of them, part gold and silver. The Assay Offices assay coins and bullion. The New York office is the only Assay Office that parts; but all the Mints part except the one in Denver.

The processes which are used in the United States are: the sulphuric, which is used everywhere but at Philadelphia, where the nitric acid process is used, and San Francisco, where both processes are used.

The material which comes into the Mints and Assay Offices of the United States, consists of gold and silver bricks, coins, plate, old jewelry, amalgam, and, in fact, anything containing gold or silver, or both. Formerly a great deal of Dore bullion was received from the Nevada mines, most of which contain as high as 5 or 6 per cent. of gold. Occasionally material containing as high as 15 to 65 per cent. of lead is received. All this material is taken on deposit, and no matter of what kind it may be, each deposit is melted up by itself and is then assayed, its value affixed to it, after which

it is turned over to the parting-house. After the gold and silver deposits are sent to the parting-house to be parted they are no longer kept separate. The materials containing lead are put on one side and are several times a year cupelled on an English iron water-back cupel hearth. It is usually arranged that this work of cupellation shall be done during the winter, as the heat of the process in the confined space where the furnace is situated would be too great in the summer months.

When the deposit is made the depositor receives a receipt stating the weight of the metal in Troy ounces and decimals of an ounce. No value is attached to the receipt. This value is subsequently determined by assay. The bullion so received is placed in a wooden box with a label stating its history, and a number is chalked on the lid. After the melting this number is stamped on the bullion produced. The only object of this preliminary melting is to make the metal homogeneous, so that an assay may be made of it. After melting the weight is again taken so as to ascertain the loss in melting. When the deposits are large enough they are cast in what is called a shoe mould. This mould is made of cast iron; it is 13 × 3 inches, and 1½ inches deep. It has two separations in it, which divide the bar into three pieces. A silver deposit bar from this mould would weigh about 200 ounces.

A twenty horse-power engine does the work of crushing the crucibles, debris of furnaces, etc. The melting-room for the deposits is a separate department, and contains five furnaces for the purpose. These furnaces require the attendance of four men. When, however, the deposits are received faster than these furnaces can take them the furnaces in the parting-house are used for melting. The accounts of this department are kept entirely separate from the parting establishment, and whatever bullion goes out of it is charged to the parting.

The following table gives the deposits and purchases of gold and silver bullion at the Mints and Assay Offices of the United States for the fiscal year ending June 30th, 1884:

* Description.	Mints.					Assay Offices.					Total.
	Philadelphia.	San Francisco.	Carson.	Denver.	New Orleans.	New York.	Boisé.	Helena.	Charlotte.	Saint Louis.	
GOLD.											
United States bullion (domestic production)	$193,653 13	$17,966,265 39	$1,451,819 72	$1,360,655 54	.....	$7,061,188 89	$169,683 52	$734,269 64	$131,362 45	$10,698 05	$29,079,596 33
United States coin	89,704 97	4,922 90	.....	.....	$13,912 83	152,273 94	.....	.....	17 30	2,285 23	263,117 17
Foreign bullion	66,058 29	1,763,026 17	.....	.....	12,702 85	4,181,586 88	.....	.....	114 18	246 08	6,023,734 45
Foreign coin	15,264 45	3,566,641 06	.....	.....	29,564 89	5,476,076 78	.....	.....	58 01	7,856 26	9,095,461 45
Jewelers' bars, old plate, etc.	639,504 65	18,674 16	.....	3,435 83	35,343 15	1,134,689 10	.....	.....	952 72	32,169 65	1,864,769 26
Total	1,004,185 49	23,319,529 68	1,451,819,72	1,364,091 37	91,523 72	18,005,815 59	169,683 52	734,269 64	132,504 66	53,255 27	46,326,678 66
Redeposits { Fine bars	1,013 28	.....	.....	.....	.....	1,381,765 53	.....	.....	.....	.....	1,382,778 81
Redeposits { Unparted bars	1,661,709 45	98,418 48	.....	3,509 73	.....	883,377 19	296 11	.....	.....	.....	2,647,310 96
Total gold	2,666,908 22	23,417,948 16	1,451,819 72	1,367,601 10	91,523 72	20,270,958 31	169,979 63	734,269 64	132,504 66	53,255 27	50,356,768 43
SILVER.											
United States bullion (domestic production)	12,034,081 38	4,849,521 13	958,639 90	19,426 27	8,063,630 93	5,512,480 21	2,026 61	22,439 18	750 28	117 99	31,463,113 88
United States coin	137,339 38	2,163 95	92 51	.....	10,083 93	2,219 39	.....	.....	130 28	1 76	152,031 20
Foreign bullion	1,251 27	2,065,000 84	.....	.....	53,607 01	404,313 50	.....	.....	.....	569 91	2,524,742 53
Foreign coin	916,635 61	25,412 69	.....	.....	956,078 22	86,230 07	.....	.....	14 03	.....	1,984,365 62
Jewelers' bars, old plate, etc.	127,632 44	2,646 57	.....	.....	32,961 16	228,116 84	.....	.....	145 02	4,535 10	896,037 13
Total	13,216,940 08	6,944,745 18	958,732 41	19,426 27	9,116,356 25	6,233,360 01	2,026 61	22,439 18	1,039 61	5,224 76	36,520,290 36
Redeposits { Fine bars	897,767 46	94,169 73	.....	.....	.....	37,950 10	.....	.....	.....	.....	1,029,887 29
Redeposits { Unparted bars	21,134 64	1,262 07	.....	.....	.....	25,804 73	6 40	.....	.....	.....	48,207 84
Total silver	14,135,842 18	7,040,176 98	958,732 41	19,426 27	9,116,356 25	6,297,114 84	2,033 01	22,439 18	1,039 61	5,224 76	37,598,385 49
Gold and silver deposits and purchases	14,221,125 57	30,264,274 86	2,410,552 13	1,383,517 64	9,207,879 97	24,289,175 60	171,710 13	756,708 82	133,544 27	58,480 03	82,846,969 02
Redeposits { Gold	1,662,722 78	98,418 48	.....	3,509 73	.....	2,265,142 72	296 11	.....	.....	.....	4,030,089 77
Redeposits { Silver	918,902 10	95,431 80	.....	.....	.....	63,754 83	6 40	.....	.....	.....	1,078,096 13
Total gold and silver	16,802,750 40	30,4 8,125 14	2,410,552 13	1,387 027 87	9,207,879 97	26,568,073 15	172,012 64	756,708 82	133,544 27	58,480 03	87,955,153 92

* Report of the Director of the United States Mint for 1884, p. 35.

The following table gives the work of the United States Mints and of the New York Assay Office. It shows the material operated upon, and the wastage during the years 1883 and 1884:

*PHILADELPHIA.		1883.	1884.
Deposits	value...	$15,319,070 79	$16,802,750 40
Gold coinage	pieces...	941,680	425,334
Silver coinage	do. ...	18,798,076	19,406,793
Minor coinage	do. ...	60,951,526	55,955,029
Total coinage	do. ...	80,691,282	75,787,156
Gold coinage	value...	$7,729,982 50	$2,777,154 00
Silver coinage	do. ...	12,325,470 15	13,854,387 80
Minor coinage	do. ...	1,428,307 16	1,174,709 73
Total coinage	do. ...	$21,483,759 81	$17 806,251 53
Gold bars	do. ...	$381,508 84	$521,095 65
Silver bars	do. ...	280,174 78	54,282 11
Total bars	do. ...	$661,683 62	575,387 76
Gold operated upon by melter and refiner	standard ounces	1,231,102	764,264
Silver operated upon by melter and refiner	do.	23,403,427	26,370,821
Gold operated upon by coiner	do.	979,842	541,311
Silver operated upon by coiner	do.	22,187,911	25,503,602
Gold wastage of melter and refiner	do.	.....	18
Silver wastage of melter and refiner	do.	.....	.....
Gold wastage of coiner	do.	22	15
Silver wastage of coiner	do.	4,952	6,092
SAN FRANCISCO.			
Deposits	value...	$35,328,140 73	$30,458,125 14
Gold coinage	pieces...	1,357,000	1,294,450
Silver coinage	do. ...	7,350,000	5,300,000
Total coinage	do. ...	8,707,000	6,594,450
Gold coinage	value...	$26,760,000 00	$23,543,500 00
Silver coinage	do. ...	7,350,000 00	4,850,000 00
Total coinage	do. ...	$34,110,000 00	28,393,500 00
Gold bars	do.	.....	$5,450 64
Silver bars	do. ...	$1,246,779 85	1,234,640 06
Total bars	do. ...	$1,246,779 85	$1,240,090 70
Gold operated upon by melter and refiner	standard ounces	2,671,989	2,323,092
Silver operated upon by melter and refiner	do.	13,974,639	10,903,752
Gold operated upon by coiner	do.	2,651,370	2,329,181
Silver operated upon by coiner	do.	12,628,737	9,779,063
Gold wastage of melter and refiner	do.	.....	.....
Silver wastage of melter and refiner	do.	16,573	16,309
Gold wastage of coiner	do.	134	120
Silver wastage of coiner	do.	638	618

* Report of the Director of the United States Mint for 1884.

CARSON.		1883.	1884.
Deposits	value...	$2,766,200 93	$2,410,552 13
Gold coinage	pieces...	104,042	107,955
Silver coinage	do. ...	1,120,000	1,164,000
Total coinage	do. ...	1,224,042	1,271,955
Gold coinage	value...	$1,403,945 00	$1,612,170 00
Silver coinage	do. ...	1,120,000 00	1,164,000 00
Total coinage	do. ...	$2,523,945 00	$2,776,170 00
Gold bars	do.	.....	.....
Silver bars	do. ...	17,131 66	.....
Total bars	do. ...	17,131 66	.....
Gold operated upon by melter and refiner	standard ounces	148,207	186,893
Silver operated upon by melter and refiner	do.	2,553,537	2,241,395
Gold operated upon by coiner	do.	160,977	172,920
Silver operated upon by coiner	do.	2,039,355	2,194,509
Gold wastage of melter and refiner	do.	92	71
Silver wastage of melter and refiner	do.	660	1,833
Gold wastage of coiner	do.	6	6
Silver wastage of coiner	do.	366	472

NEW ORLEANS.			
Deposits	value...	$8,776,566 66	$9,207,879 97
Gold coinage	pieces...	4,300	.....
Silver coinage	do. ...	8,040,000	8,905,000
Total coinage	do. ...	8,044,300	8,905,000
Gold coinage	value...	$43,000 00	.....
Silver coinage	do. ...	8,040,000 00	$8,905,000 00
Total coinage	do. ...	$8,083,000 00	$8,905,000 00
Gold operated upon by melter and refiner	standard ounces	12,675	.....
Silver operated upon by melter and refiner	do.	13,849,995	16,301,242
Gold operated upon by coiner	do.	6,378	.....
Silver operated upon by coiner	do.	13,073,130	15,146,418
Gold wastage of melter and refiner	do.	6	.....
Silver wastage of melter and refiner	do.	1,656	9,089
Gold wastage of coiner	do.	.....	.....
Silver wastage of coiner	do.	566	1,708

NEW YORK ASSAY OFFICE.			
Gold deposits	value...	$17,016,061 56	$20,270,958 31
Silver deposits	do. ...	6,592.657 50	6,297,114 84
Total deposits	do. ...	$23,608,719 06	$26,568,073 15
Gold, fine bars manufactured	do. ...	$11,678,319 07	$13,161,450 71
Gold, mint bars manufactured	do. ...	2,195,293 37	5,727,797 27
Gold, standard bars manufactured	do. ...	787,543 24	1,388,946 66
Gold, sterling bars manufactured	do. ...	1,325,191 05	613,234 34
Silver, fine bars manufactured	do. ...	6,433,097 08	6,081,756 65
Silver, standard bars manufactured	do. ...	15,312 58	37,163 64
Silver, sterling bars manufactured	do.	.....	.....
Silver, mint bars manufactured	do.	.....	181,719 13
Total bars manufactured	do. ...	$22,434,756 39	$27,192,068 40

*The number of gross ounces of bullion refined or parted in the acid refineries, and the ounces at standard fineness and values of the gold and silver resulting from those operations, are shown in the following table:

OUNCES.

Mint or Assay Office.	Gross.	Standard Gold.	Standard Silver.
Philadelphia	879,090.18	135.577,210	789,425.03
San Francisco	5,505,896.55	499,452,009	5,134,621.65
Carson	1,189.289.30	88,484,848	1,142,713.74
New Orleans	52,604.28	2,897,851	51,211.74
New York	5 514,176 00	617,517,000	4,639,779.00
Total	13,141,056.31	1,343,928,918	11,757,751.16

VALUE.

Mint or Assay Office.	Gold.	Silver.	Total.
Philadelphia	$2,522,366 70	$918,603 67	$3,440,970 37
San Francisco	9,292,130 40	5,974,832 46	15,266,962 86
Carson	1,646,229 73	1,329,703 26	2,975,932 99
New Orleans	53,913 51	59,591 84	113.505 35
New York	11,488,688 37	5,399,015 56	16,887,703 93
Total	$25,003,328 71	$13,681,746 79	$38,685,075 50

The following tables give the expenses and earnings of the United States Mints and Assay Offices for the year 1884:

Institution.†	Salaries.	Wages.	Contingent.	Coinage of the standard silver dollar, Act of February 28, 1878 (indefinite).	Total.
Philadelphia Mint	$40,503 18	‡$293,229 91	§$100,173 35	$84,422 70	$518,329 14
San Francisco Mint	43,400 00	231,855 75	54,650 07	4,649 00	334,554 82
Carson Mint	29,050 00	53,999 25	22,134 17	17,308 54	122,491 96
New Orleans Mint	31,873 10	73,988 24	34,998 00	29,674 54	170,533 88
Denver Mint	10,950 00	12,000 00	5,507 84	.....	28,457 84
New York Assay Office	39,250 00	23,038 50	8,685 16	.....	70,973 66
Helena Assay Office	7,950 00	11,389 55	7,998 93	.....	27,338 48
Boisé Assay Office	3,000 00	.....	4,973 34	.....	7,973 34
Charlotte Assay Office	2,750 00	.....	1,973 95	.....	4,723 95
Saint Louis Assay Office	3,500 00	.....	1,838 51	.....	5,338 51
Total	212,226 28	699,501 20	242,933 32	136,054 78	1,290,715 58

* Report of the Director of the U. S. Mint, for 1884.
† Report of the Director of the United States Mint, for 1884, p. 11.
‡ Includes $229.91 expended from reimbursements by Hawaiian Government for coinage dies.
§ Includes $173.35 from reimbursements for materials used.

The amount collected at the various Mints and the New York Assay Office of depositors and sellers of bullion during the fiscal year, as charges for parting, refining, and toughening bullion, amounted to $268,489.33. These charges are by law required to be used in paying the expenses of the operations, for which purpose the sum of $247,798.79 was expended at the institutions named during the year.

The following statement shows the amount collected and expended for this purpose at each institution :

Institutions.	Charges collected.	Expenditures.
Philadelphia Mint	$11,404 45	$10,309 27
San Francisco Mint	134,106 25	132,579 53
Carson Mint	16,630 97	21,803 33
New Orleans Mint	738 63	34 77
New York Assay Office	105,609 03	83,071 89
Total	$268,489 33	$247,798 79

The balance remaining in the Treasury of the United States at the close of the fiscal year 1884 to the credit of the parting and refining appropriation, being the excess of the charges collected for those operations over the expenditures from the 1st of July, 1876, to June 30, 1884, was $182,484.83.*

In the year 1881–82 the New York Assay Office parted 5,139,768 ounces of bullion, of which 507,533 ounces were fine gold. There were, therefore, dissolved in acid 4,632,235 ounces.

Amount treated,	5,139,768	ounces.
Treated with fluxes and fire,	1,522,515	"
	6,662,283	"
Standard weight of gold,	563,926	"
Standard weight of silver,	6,079,555	"
	6,643,481	"

The process used in the New York Assay Office is known as the sulphuric acid process. It consists of eight operations :

1. Inquartating and granulating the bullion.

* Report of the Director of the Mint, for 1884.

2. Solution of the granulated bullion in sulphuric acid.
3. Sweetening and fusion of the insoluble residue of gold.
4. Condensation of the acid fumes.
5. Reduction of the sulphate of silver by copper.
6. Sweetening and melting of the precipitated silver.
7. Treatment of the sulphate of copper.
8. Treatment of the debris of furnaces, sweepings, etc.

1. Inquartating and Granulating the Bullion.

When the bullion is turned over to the parting-house they recognize several varieties of it, which are carefully separated so as to treat them in the various mixtures. All the material received is first melted and inquartated so as to have two of silver and one of gold, or three to four of silver to one of gold, the attainment of a high fineness of gold depending on the amount of copper. The charge for melting is made so that the copper does not exceed one-twelfth part of the entire weight of the metal. It is generally preferable not to have it above 6 per cent. It is usual to adjust the gold, when good silver is plenty, so that it will contain about 6 per cent. of copper and 75 of silver. In any case the relation of the gold and the copper must be not over one-third and one-twelfth respectively of the entire weight of the bullion. When the copper contains not more than 8 per cent. of silver and no gold, it is melted with the reducing-house scrap, and used to precipitate the silver, or melted with similar silver of a higher fineness, and used in making standard bars. When it contains gold without silver it is used to dilute silver containing too little gold, or kept to make standard gold. If it contains both it is treated like any other deposit.

The gold, especially that which comes from California and the Frazer's River district, often contains a considerable quantity of platinum and the platinum metals. This is a matter of great inconvenience, for when iridosmium is found in the jewelers' bars it makes a great deal of trouble. It is always desirable to separate it, and in order to do it where it is suspected the cupels and cornets of the assay are carefully watched for the dark specks. If not detected here it shows itself in the hard dark spots in the manufactured article. It is always separated when found. This is done by first ascertaining how much platinum and iridosmium are present by an assay. It is then melted with from four to five times its weight

of pure silver, so as to make at least three to four parts of silver to one of gold. The more silver the better, as it increases the difference in the density of the metals. When fluid it is left in a liquid condition quiet for half an hour. It is then poured off, and the platinum metals by their gravity settle to the bottom. As soon as in pouring the metal becomes thick the pouring is stopped. The crucible is put back into the furnace and the heat brought up so that the material accumulated on the sides will melt. In this way a king is formed at the bottom. This is allowed to stand in the fire over night, and is then assayed. If gold is found in it, it is melted again with two to three times its weight in silver, and the same process repeated. Usually after the second operation it will be found that there is no appreciable quantity of gold in it. The platinum metals will thus be found entirely separate from the gold and in a considerable quantity of silver. This is accumulated in the vaults as metal containing platinum. Once in a long period of time it is dissolved in sulphuric or nitric acid. The amount is not very large, about 150 ounces only having accumulated in about ten years. The separation is never done until it has accumulated sufficiently to make it worth while.

The crucibles used are of graphite. They will usually last six melts of six to seven hours each. After this they either crack or are worn so thin that it is no longer safe to use them. The charge for a granulation silver melt is always 160 pounds. The furnace is the ordinary one with an inclined sliding door, heated with anthracite coal. When the melted charge is nearly ready to granulate the cover of the crucible is removed and a small cup of burned fire-clay made for the purpose, holding about one quart, is put on the top of the melted mass to be brought up to a red heat. When it is ready for granulation this cup is seized by a special iron tongs and the metal thrown with a peculiar rotary motion, which has for its object to make the metal strike the water in different places, and in a flat stream, so that the granules will be thin and flat and not lumpy or spherical, and thus expose to the acid the greatest possible surface. Slight changes in the manipulation make very decided ones in the result. It requires a considerable amount of dexterity to throw the stream of metal so as to give the granules the desired shape. The vessel into which the metal falls is made of copper, and is thirty inches in diameter and the same in depth. Inside of this, fitting the vessel loosely at the bottom, is a round

copper pan 5 inches deep, with three handles on the side, so arranged that when lifted out by a Weston pulley-block attached to a swinging crane it will tip to one side, where there are holes in the bottom. It is raised to the top of the cylinder, set down on it, and left there to drain. When the water ceases to run from it, it is carried by the crane to an inclined cast-iron pan 43 inches long by 27 inches wide and 4 inches deep, having flaring sides. Into the bottom of this pan a piece of gas-pipe is screwed, which conducts all the water into a wooden box lined with copper. The water draining off always carries with it some small pieces of the granules. The granules are left in the iron basin to dry, and are then stored in wooden boxes lined with copper having two iron handles on the outside. The boxes are made of 1-inch pine, 18 inches long, 11½ inches wide, and 6 inches deep. These are piled one on the top of the other until ready for solution. The amount granulated at a time is from 150 to 160 ounces. The melting lasts from six to seven hours every day. The granulations are made about once an hour, so that each pot lasts about thirty-six heats. The pot used in melting for the granulation is a No. 70 graphite pot. Two or three will generally be used up in a week's work. Two men do all the work of granulation. There are two granulation furnaces which, with the crane and granulating tank, are together. When there is any necessity, however, any one of the other furnaces may be used for this purpose. The granulations when drained are placed in wooden boxes until ready to be dissolved.

In some of the establishments, as at Philadelphia, great care is taken to have the water as cold as possible, ice being put into it when it is not sufficiently cool. In others, special care is taken to have the water in constant motion. In others the water is made to flow in constantly, the overflow being carried off by a special pipe. In others the water is allowed to remain at the temperature of the air, but is drawn off with a syphon after every operation.

2. Solution of the Granulated Bullion in Sulphuric Acid.

The granulated bullion is treated in a series of cast-iron pots. Three of these are of large size and two are small. The large sized pots are placed upon one side of the room and the small ones on the other, both having fire-places underneath them. They are heated with anthracite coal. The dimensions of the larger

furnaces are 45 inches in diameter, 43 inches deep, with a rim 3 inches wide. Each is set in a cast-iron plate 64 inches square and 55 inches from the ground. They weigh 1100 to 1200 pounds each, and hold about 160 gallons of acid. The smaller pots are 24 inches in diameter and 26 inches deep. They weigh about 850 pounds and hold about 50 gallons of acid. They are set in a cast-iron frame 53 inches by 48 inches and 48 inches above the ground. These furnaces are built with the ash-pit above the floor, which makes them a little too high for convenience, as the top has to be reached by steps. In the new constructions they are made 36 inches high, and the ash-pit is sunk below the floor. These are both covered with hoods, which are made of 8-pound lead, which connect by a pipe 10 inches in diameter having a hydraulic seal, so that it can easily be taken down when necessary, with the condensing apparatus. Each of these hoods has two openings 17 by 26 inches for the introduction of the alloy and for working in the pot. These are closed by sheets of lead, which are simply set against them with a part of the cover turned over at the top so as to allow of their being easily removed. Two of the largest pots are used for the solution of the silver granules. The charge is 450 pounds, over which from two to three times the weight of the acid at 66° B. is put. This is for the ordinary alloy. When the bullion is 860 fine a charge of 450 pounds may be made, but when it is only 850 fine the charge is reduced to 400 pounds. A copper charge with only 55 per cent. of silver has been worked directly. In the year 1884, 300,000 to 400,000 ounces of this kind of bullion with only 55 per cent. silver, 45 per cent. of copper, was treated directly with acid, but the charge was only 250 pounds. The lowest ever treated before was 76 per cent. silver to 24 per cent. copper. The usual charge is about 300 pounds of granules. The charge of acid is made in two successive additions, the first charge being about 650 pounds. In some places when there is very little gold in the silver the charge is sometimes made much greater, as the amount of gold remaining behind in the pot would be too small to treat. The amount of gold in this office is, however, almost always about the same. If there is any considerable amount of copper in the granules the charge has to be smaller, as the copper requires more acid than the silver.

The charge once in the pot, the acid is gradually raised to the boiling point and kept there for several hours during which time 250

to 300 pounds of acid is gradually added. When the solution is complete the sulphate of silver is drawn off by means of a siphon and run into vats on the floor below. Considerable care is required in performing this operation in order not to carry over any gold, and to effect this the gold is carefully levelled over the bottom of the boiler, a lead siphon with the end turned up about three inches is then introduced to draw off the supernatant liquor. The fire is then withdrawn and fresh acid is placed over the gold which is then ladled out with an iron ladle perforated with holes and placed in a cast iron scoop which fits into one of the holes in the conical cover of the pot. This scoop for convenience of handling has three handles, one on the end and one on each side. Almost every particle of gold is in this way drawn out of the pot. From the scoop it is transferred to the smaller pots each one of the pots receiving the charge from one of the large ones. Occasionally in removing the gold from the pots some sulphate of silver insoluble in the concentrated liquors is thrown down with it. This is put one side to be treated by hot water and acid when it is convenient. The amount of gold ladled over into the smaller pots will be not far from one hundred pounds. The gold is boiled with about two hundred pounds of acid in these pots for from 2½ to 3 hours. This acid is then removed and it is boiled again with a fresh supply of acid. After two boilings in the small boilers the acid is removed and the charge in both of them is put into the third of the large ones and boiled there twice. This second boiling takes another day, so that the granules will have received one boiling lasting a whole day in the large pots, two boilings of the gold residues in the small pots and one boiling of the accumulated gold residues in the large pots, making five boilings lasting over three days. The gold in this way is made 997½ fine. Another boiling would easily bring it up to 998 or even higher, but this is not now necessary. The metal with the gold is silver. When the gold was in considerable quantities in the bullion received the first solution was made in nitric acid, and a single subsequent solution was made in sulphuric acid as now, which uniformly gave gold 998 fine. This process, known as the Mason process, was invented by Mr. Andrew Mason, the present Superintendent of the N. Y. Assay Office, in the year 1866, at which time he was the melter and refiner, and resulted in very greatly reducing the expenses of parting. As soon as the quantity of gold became large it was abandoned for the present

process. Four men are required for the work of these furnaces. They do all the work of transfer, washing and sweetening.

The amount and cost of the acid used in the fiscal year ending June 30th, 1882, is given below:

Sulphuric acid, 66° B., .	1,579,340 lbs. at $1.25, =	$ 19,741.73
Sulphuric liquor, 40° B., .	2,108,570 " " 25cts. =	5,271.17
Net cost, ...		14,470.56
Less 200bbls. blue vitrol exchanged, ...		4,266.11
		$10,204.45

3. SWEETENING AND FUSION OF THE INSOLUBLE GOLD RESIDUE.

After the gold has been boiled it is collected in an iron scoop, emptied into a lead-lined tub where it is washed several times by decantation, and then transferred to a wooden vat upon wheels, lined with 6-pound lead, where the sweetening is completed. This tank is 28 by 33 inches and 17 inches deep. A wooden filter filled with holes is placed upon its bottom, the top of which is 4 inches above the bottom. The bottom is made to incline towards the center and from the lowest point there is a pipe which is made to communicate with the reservoir into which the liquid is to run. The wooden filter is carefully covered with a cloth and the gold is put upon this cloth and washed once with cold and twice with hot water until it is entirely sweet. This takes from two to three hours. When the gold is quite sweet it is pressed into cakes by a hydraulic press. These cakes are twelve inches in diameter and three inches thick, and weigh from 130 to 150 pounds. The cakes of gold are dried in an oven heated with steam. They are then broken up and melted in graphite crucibles and fluxed with a cover of bone ash into which nitre is introduced as in the silver melting. The material remaining in the furnace from three to five hours is then cast into bars which run from 997.5 to 998.5 fine.

The charge for a gold melt is 6000 ounces, a No. 70 graphite crucible being used. The casting is done in the same way as the silver. The moulds are of iron with a cover and are made of various sizes to suit the demand for bricks of various values. Those for bricks which will have a value of about $8000 are 7 by 3½ inches, by 1¾ inches. Those worth about $5000 are 6 by 2¾ by 1⅜ inches. Those which are 2¾ × 1½ × 1 hold about 32 ounces; the brick is worth $640 to $650. The size 2¼ × 1½ × 1

holds 20 ounces; the brick is worth above $400. The size 2 × 1¼ × ½ holds about 15 ounces, and the size 1¾ × 1 × ½ holds 10 ounces. These bricks are all stamped with the stamp of the Office, and pass current in exchanges. Only one furnace is specially designated for the melting of the gold. Any other one might be used and often is when the work presses.

4. Condensation of the Acid Fumes.

The acid from all the extra boilings is collected in two cast iron pots of the same size as the large solution pots on the same floor; but on a lower level than the bottom of the boiling pots. The acid is siphoned out into an iron launder and runs in to these pots. The top of these pots is covered with sheet lead so as to prevent the escape of any acid fumes. The two are connected by means of a curved pipe and from the center of the second one a pipe ten inches in diameter, of 8-pound lead, goes to the condensing apparatus. The acid which is used is brought to the establishment in large iron tanks from which it is discharged into an acid egg in the cellar. From there it is pumped into tanks in the attic where are also the water supply tanks for the entire establishment. The acid is conducted in pipes and the quantity which is required is added by means of stop cocks, it not being necessary to touch the acid at any time. All the acid fumes are carried first into lead chambers which are made of 10-pound lead. These chambers are 5 by 10 feet and 7 in height and are supported on the outside by a wooden frame. About the middle of this chamber ¾ inch iron pipes covered with lead about an inch and a half apart are placed. These pipes are supported on pieces of wooden joists also covered with lead and on top of these 30 inches of the best Connellsville coke is placed, the coke being supplied with a constant stream of water. From the top of the vat the gases pass through four vertical pipes 16 inches in diameter, made of 10-pound lead, in which there is a spray of water from a rose. From here they pass to a central circular chamber, also made of lead and supported with wood, which is 8 feet 6 inches high by 6 feet in diameter. This has a coke packing in the center and from this they pass through a series of 20 inch pipes 22 feet high in which there are a constant series of water sprays. Generally this condensation is quite sufficient for the purpose. There are a double set of these condensers, but when for any reason the condensation is not sufficient the

nitric peroxide formed by boiling saltpeter in sulphuric acid is added to the first chamber to collect and condense the gases. All the men engaged on the acid and crystallization works are obliged to wear long rubber gloves and boots to protect their hands and feet.

5. Reduction of the Sulphate of Silver by Copper.

All the water used in sweetening the gold is collected in two tanks on the floor below. They are 7 feet long, 5 feet wide on the top, 4 feet wide on the bottom, and 3 feet 10 inches deep. They are lined with 8-pound lead. The bottom is not made level, and their contents can be discharged into the reducers into which the acid sulphate of silver runs from the solution pots. There are four of these reducers, which are 14 feet long, 7 feet wide, and 30 inches deep. They connect with the stack by a flue. The top is covered with an inclined roof which rises 10 inches above the level of the sides. These tanks are all heated by live steam. It is discharged directly into them by a vertical lead pipe 1½ inches in diameter, which goes to within 2 to 3 inches of the bottom of the tank. They were formerly heated by coils, but a burst in the pipe before the entrance of the steam into the coils, showed that the liquor could be heated just as well with live steam, and since that time it has been used directly. There is a stopcock on the pipe for regulating the amount of steam.

Before the introduction of the acid liquors, copper plates 10 by 21 inches and 1 inch thick are stood around all the sides, leaning against them, and are laid flat upon the bottom. The object of the plates is to reduce the silver. The acid sulphate liquor being allowed then to flow into the tank, is diluted by the wash water to about 20° B. When the tank is full it is brought up to violent ebullition with steam and kept so for five or six hours. The silver is precipitated by the copper, and is removed from the tank every other day, two tanks being emptied each day. The copper of the plates is transformed into copper sulphate. They are never taken out until they become too thin or too small to be of any further service. During the precipitation the liquor is constantly stirred with wooden stirrers. The solution is first tested for silver, and is allowed to settle over night. In the morning it will have completely settled. It is again tested to see whether any silver is present. It sometimes happens that the liquor takes up silver over night, so that it must not only be tested

at night, but again in the morning. The consumption of copper in precipitating the silver is very much reduced by the melting of the silver deposits which do not contain gold and are under 400 fine, with the copper used for precipitation. The silver is thus obtained pure without a direct parting, and the copper alloy takes the place of that amount of copper that would otherwise have to be purchased for it. Occasionally there is a leak in this tank, in which case the liquor must all be removed, and then, in order to get the silver from the copper plates, it will generally be necessary to scrape them. Usually, however, this is not necessary. Whatever tools, such as hoes, shovels, etc., are used during the operation, are made of copper in order not to lose silver, as well as to prevent the liquor from becoming impure.

6. Sweetening and Treatment of the Precipitated Silver.

When the liquor tested shows that there is no longer any silver present the silver is drawn as much as possible to one side with copper hoes so as to be as far from the discharge outlet as possible. The contents of the vat is then discharged into a filter made of copper, 36 inches long, 27 inches wide, and 7 inches deep, the bottom of which is filled with 9 rows of holes ⅜ of an inch in diameter, at intervals of 2½ inches. There are also two rows of these holes of the same size on the sides. It has two handles upon the sides, and is covered with a linen cloth. The liquid contents of the vat is discharged into this filter, and from there runs into a tank. Whatever particles of silver are in the liquor are retained by the filter. The silver is sweetened with hot water in a lead-lined tank upon wheels arranged exactly as the gold sweetener is. These tanks were formerly made round, but that shape has now been abandoned, and the rectangular substituted for it. After the silver is entirely sweet, which will be in about two hours, it is carried in these vats to the hydraulic press, where a pressure of 200 tons is placed upon it, and it is made into cakes 1 foot in diameter and 3 inches thick, which weigh from 65 to 75 pounds. The water which is squeezed out of it runs into a trough around the table of the hydraulic press, and from there through a pipe into a linen bag, which is securely fastened to its end, and is suspended over a series of six wooden traps about 1 foot square. This is discharged into a pipe which runs into the cellar; it then flows through four other traps and from there goes to the sewer. In this way all the silver is collected.

These cakes from the press are cleaned of any grains of silver attached to them, and are then placed in the copper steam drier, which runs around the room heated as hot as live steam can make them, and are left about eighteen hours to dry. After this they are broken up and melted. For this purpose a No. 100 graphite crucible is used, the charge being from 4500 to 6000 ounces.

In melting the broken silver cakes it is always desirable to have the precious metals under a cover, and this cover is made of bone-ash, which is put over it as soon as the metal is melted. In order to refine the metals this cover is parted in the middle, and about once every half hour a handful of nitre, with some borax and soda ash, is put upon the metal. When all the nitre has been absorbed in the bone ash, it is removed and a fresh cover of bone ash is put on and nitre added again, and this is repeated ten or twelve times during the six hours of melting. The cessation of the action of the nitre is recognized by the workmen by a peculiar motion of the whole cover. The object of the nitre is to oxidize out the base metals and bring the silver up to 998 to 999 fine. In the other meltings only charcoal is used as a cover, but it is not necessary and is not always done on account of the waste of charcoal.

In front of each furnace there is a ledge on which the moulds for the silver bricks are placed. The silver in casting is dipped out of the pots exactly as in granulating; it is poured into iron moulds which have a tightly fitting cover. These moulds are of several sizes; the largest are 12 × 6 inches, and 3 inches deep. Such moulds will hold about 1000 ounces. Other moulds of the same capacity are 3½ and 4 inches deep. The next size is 8 inches long, 3½ inches wide, and 2 inches deep; this holds about 220 ounces. Smaller moulds, 5 × 2½ and 1½ inches deep, hold about 80 ounces. Those 5 × 2½ and 1½ inches deep hold 30 to 40 ounces. These are the smallest used. As soon as the metal is quiet any scum that may have risen is taken off, and when it is just about to become solid the iron cover is put on. When solid, but still red hot, it is removed from the moulds and set up on end on a cast-iron pan 48 inches long and 30 inches wide with a rim 1½ inches deep to cool. From miscellaneous mixtures it is comparatively easy to get both the gold and the silver up to 990 fine, but very difficult to get it up to 999. The base metal in the silver is copper. Seven furnaces do all the work of melting the fine silver.

7. TREATMENT OF THE SULPHATE OF COPPER.

The solution from the silver vats containing the copper is run into concentrating tanks of which there are two. The liquor which runs into these tanks is only at 20° B., and it is run into them to be concentrated. This is done by means of a lead steam pipe which has 8 turns and is 320 feet long on the bottom of the vat. When it has been sufficiently concentrated it is run into a series of seven intermediate crystallization tanks. The tanks are 15 feet long, 7 feet wide and 30 inches deep. The liquor is there brought up to from 35° to 40° B. When at that concentration it is allowed to cool, and the copper crystallizes out in two to three days. The mother liquor from these vats is first concentrated up to 40° B. and then run into storage tanks upon the outside of the building where it is stored until sold. The copper crystals which are removed from these vats are carried in wooden wagons to the solution house on the floor below. This is a tank with an inclined roof 14 feet long, 7 feet wide, 30 inches deep, the rise in the roof being 12 inches. Here it is dissolved and brought up by steam to 35° to 37° B. From here it is carried by a rubber hose to the crystallization tanks on the same floor, but below the bottom of the solution vats. There are eight of the crystallization vats. These tanks are 9 feet long, 5 feet wide at the top and 2 feet at the bottom, and are covered with 10-pound lead. Over the top of these vats are 36 pieces of wood, on each piece there are six strips of 6-pound lead 30 inches long and 3 inches wide. They are hung on a nail and fall into the vat reaching almost to the bottom; on cooling large crystals of copper sulphate are formed on them. Small ones are also formed upon the bottom and sides of the tank. About once a week or whenever it is convenient for the work of the Office, the mother liquor from the crystallization vats is pumped up into the concentration vats and the crystals removed, to do which the lead strips are taken to an inclined lead table 10 feet long, 3½ feet wide, with a rim of about 6 inches and lined with 10-pound lead. The largest crystals are always taken from the lead strips; on the sides and bottom the crystals are smaller. When there are any impurities, such as sulphate of iron, attached to any of the masses of copper sulphate, these are scraped off with an ordinary iron scraper. The crystals are washed on this table with cold water. From the tanks at some distance these crystals are brought in a wooden car 30 by 24 inches and 26 inches high, and this car is provided with a

double bottom with an outlet behind, under which a pail is set to catch any drippings. After washing all the crystals are carried to wooden bins made with boards some distance apart on the bottom, where they dry. They are then packed in barrels which weigh from 360 to 375 pounds; from 100 to 150 barrels per month of this copper are made. The total annual production of copper sulphate is about 450,000 pounds. No separation is made between the large and small crystals.

There is a great demand for the copper crystals which have acquired a certain reputation and are, therefore, easily sold. The proceeds of the sale nearly cover the cost of the acid and copper purchases. Four men working on two different floors do all the work of the reducing house, and of the crystallization of the copper. The work in each department is intermittent, so that while one is waiting they work at the other. The steam is furnished by two boilers of 60 nominal horse-power each. The power is supplied by a 50 horse-power engine. All the transportation from story to story is done by a lift. All the floors except the melting room, which is covered with iron plates, are covered with ten-pound lead. The whole building is fire proof.

The following table, for which I am indebted to Mr. A. Mason, Supt. N. Y. Assay Office, gives the detailed expenses of the materials used in parting and refining in that Office for the fiscal year ending June 30th, 1882.

The following tables give the earnings and expenditures of the U. S. Mints and Assay Offices for the year ending June 30th, 1884.

Total expenses for the year 1881–2.		Sundries for the Year 1881–2.					
		Labor.		Supplies.		Repairs.	
Acid	$10,204 45	General	$207 03	2 Meters	$350 00	Office furniture	$48 00
Coal	7,727 82	Mason	664 10	Drying-pan, etc.	164 50	Glazing	6 00
Charcoal	951 40			Under boilers	56 53	Wire rope for elevator	93 00
Tinder-wood	272 00			Addition to boilers	895 00	Scale	3 00
Borax	1,375 92			5 Tanks	500 00	Cistern	73 22
Nitrate of soda	867 06			Lumber	192 51	Pulley	30 00
Bone ash	899 46			Brushes	60 00	Speaking-tubes	54 13
Pulverized charcoal	138 88			Chisels, etc.	11 55	Safe lock	25 00
Soda ash	21 91			Copper hoes, etc.	104 40	Copper leader	28 00
Crucibles, etc.	2,609 60			Brooms	12 00	Mill	103 53
Gloves, hose, etc. (rubber)	997 48			Oil and lard	144 75		
Sheeting and blankets	623 78			Cement	11 40		
Belting	55 78			Thread	5 62		
Iron work and mason	2,052 99			White lead	8 50		
Lead	572 88			Zinc	90 00		
Fire-bricks	157 61			Cotton-waste	12 00		
Gas	1,192 27						
Water	403 50						
Ice	72 30		$871 13		$2,618 76		$463 88
Barrels	666 00						
Labor carting ashes	836 00						
Sundries	3,953 77						
	$36,652 86						

Labor	$871 13
Supplies	2,618 76
Repairs	463 88
Total sundries	$3,953 77

*Sources.	Mints.					Assay Offices.					Total.
	Philadelphia.	San Francisco.	New Orleans.	Carson.	Denver.	New York.	Boisé.	Charlotte.	Helena.	Saint Louis.	
Parting and refining charges	$11,404 45	$134,106 25	$738 63	$16,630 97	.....	$105,609 03	.....	.....	.....	.....	$268,489 33
Melting alloy, and bar charges	636 20	2,716 45	1,145 16	292 58	.....	3,522 95	.....	.....	.....	.....	8,313 34
Charges for assaying, melting and stamping	.....	.....	.....	.....	$1,778 79	.....	$214 92	$223 27	$943 23	$80 19	3,240 40
Profits on standard silver dollars coined	1,852,718 66	697,579 80	1,247,669 69	168,704 51	.....	.....	.....	.....	.....	.....	3,966,672 66
Profits on subsidiary silver coined	92,798 46	9,852 69	.....	.....	.....	.....	.....	.....	.....	.....	102,651 15
Profits on minor coins manufactured	982,785 66	.....	.....	.....	.....	.....	.....	.....	.....	.....	982,785 66
Profits on sale of medals and proof coins	2,502 17	.....	.....	.....	.....	.....	.....	.....	.....	.....	2,502 17
Receipts from assays of ores and bullion	482 00	111 50	54 00	186 00	204 00	563 50	192 00	332 33	1,180 00	285 00	3,590 33
Deposit melting room fluxes and sweeps	2,052 61	1,006 83	425 60	287 64	975 33	4,748 77	94 49	19 45	457 76	75 32	10,143 80
Surplus bullion returned by melter and refiners	1,466 65	15,896 89	†2,527 76	.....	.....	40,472 94	.....	.....	.....	.....	60,364 24
Gain on bullion shipped the mint for coinage	.....	.....	.....	.....	1,908 70	.....	249 86	536 08	1,278 38	236 31	4,209 33
Proceeds of sale of old material	86 03	2,236 62	.....	65 87	.....	.....	.....	.....	.....	.....	2,388 52
Profits on leady melts sold	.....	.....	.....	.....	.....	1 71	.....	.....	.....	.....	1 71
Charges received from Hawaiian Government	250 00	17,500 00	.....	.....	.....	.....	.....	.....	.....	.....	17,750 00
Total	$2,947,182 89	$881,007 03	$1,252,560 84	$186,167 57	$4,866 82	$154,918 90	$751 27	$1,111 13	$3,859 37	$676 82	$5,433,102 64
EXPENDITURES AND LOSSES.											
Salaries of officers and clerks	$40,503 18	$43,400 00	$31,873 10	$29,050 00	$10,950 00	$39,250 00	$3,000 00	$2,750 00	$7,950 00	$3,500 00	$212,226 28
Wages of workmen and adjusters	358,845 70	236,504 75	93,750 95	70,310 75	12,000 00	23,038 50	3,646 68	312 75	11,389 55	1,382 50	811,182 13
Contingent expenses, not including amount paid for wastage and loss on sweeps sold	118,980 26	52,175 63	44,909 83	22,302 95	5,507 84	8,685 16	1,326 66	1,661 20	7,998 93	456 01	264,004 47
Parting and refining expenses, not including for wastage and loss on sweeps sold	103,09 27	129,890 75	.....	20,845 47	.....	83,071 89	.....	.....	.....	.....	244,117 38
Wastages of the operative officers	6,713 05	19,218 57	10,913 23	3,756 63	.....	.....	.....	.....	.....	.....	40,601 48
Loss on sweeps of the year	3,031 13	6,384 26	1,894 25	.....	.....	5,290 48	.....	.....	.....	.....	16,599 12
Loss on bullion shipped to the mint	.....	.....	.....	.....	55 82	.....	.....	.....	.....	41 66	97 48
Expenses of distributing silver coins	29,131 69	15,998 57	11,052 69	1,640 90	.....	.....	.....	.....	.....	.....	57,823 85
Expenses of distributing minor coins	29,152 32	.....	.....	.....	.....	.....	.....	.....	.....	.....	29,152 32
Minor coinage metal wasted	198 22	.....	.....	.....	.....	.....	.....	.....	.....	.....	198 22
Total	$596,864 82	$503,572 53	$194,393 05	$147,906 70	$28,513 66	$159,336 03	$7,973 34	$4,723 95	$27,338 48	$5,380 17	$1,676,002 73

* Report of the Director of the United States Mint for 1884, p. 42.

† Bullion belonging to previous years.

8. Treatment of the Débris of Furnaces, Sweepings, etc.

All the old furniture and woodwork of the buildings and the old carpets, etc., are burned, and their ashes with the worn-out crucibles, the débris of furnaces, flux covers, the sweepings of the floors and chimneys and any other material which may contain gold or silver that has accumulated about the works are treated. This is put through first a small Blake's Crusher, and then through a coffee mill grinder. From here it is accumulated in a receiver which connects by pipes with a series of funnels placed upon shelves. There were, originally, four shelves with four funnels in a row. The upper row only now contains four, the three below only two. The ends of the top ones are bent so as to reach the two funnels on the shelf below which receive the overflow of the two above. The others below discharge each into one below it. These funnels have cones 8 inches in diameter, 8 inches high and a pipe 8 inches long attached to them. Into these funnels iron tumblers 4 inches in diameter by 6 inches high are placed. The material mixed with water is allowed to discharge into the tumblers of these funnels. The overflow from the tumblers above, carrying the fines with it, goes into the three sets of two funnels below, and from there is discharged into a series of four cast iron pans 9 inches wide, 18 inches long and 4 inches deep, whose bottom is covered with mercury. From the middle of the upper of these pans there is a discharge pipe 1¼ inches in diameter, from the next below there is a pipe on each side 1 inch in diameter, and so on through the four. Each of the pans has a discharge cock at the side to draw off the mercury which covers the bottom of the pan when the pan is to be cleaned. The overflow pipe is above the level of the mercury, the discharge pipe below it. From the lower pan the material discharges into two barrels, and what settles there is dried in two copper driers, which are hollow tanks surrounded with steam and are 6 feet long by 3 feet 6 inches wide and 18 inches high. When dried they are barrelled and sold to the sweep smelters. This apparatus is cleaned as soon as eight barrels of the crushed material has passed through it. This will be every week or ten days. The residues will contain from one to ten cents per pound. When the settlement sweeps are made when the accounts are turned over at the end of the year, as the work must be very hurriedly done, these sweeps will sometimes contain 20 cents a pound. All the residues remaining in the tumblers are

collected and melted in pots to a king and separated in the usual way. The amalgam after a time is collected and distilled as usual.

The following table gives the results of treating sweepings in the U. S. Mint and the Assay Office in New York.

*Losses.	Philadelphia Mint.	San Francisco Mint.	Carson Mint.	New Orleans Mint.	New York Assay Office.	Total.
Melter and refiner's gold wastage	$338 30	.....	$1,353 10	.....	.....	$1,671 40
Coiner's gold wastage	282 43	$2,238 11	117 31	.....	.....	2,637 85
Melter and refiner's silver wastage	.....	16,360 42	1,833 27	$9,135 63	.....	27,329 32
Coiner's silver wastage	6,092 32	620 04	472 95	1,777 60	.....	8,962 91
Loss on sale of sweeps	3,031 13	5,951 39	.....	.....	$5,290 48	14,273 00
Total	$9,744 18	$25,169 96	$3,756 63	$10,913 23	$5,290 48	$54,874 48
Paid as follows:						
From contingent appropriation	.....	$2,474 74	$828 22	.....	.....	$3,302 96
From parting and refining appropriation	.....	2,255 91	957 86	.....	.....	3,213 77
From surplus bullion	$1,641 03	16,903 72	287 64	$258 46	$5,290 48	24,376 38
From silver profit fund	8,103 15	3,585 59	1,682 91	10,659 77	.....	23,981 42
Total	$9,744 18	$25,169 96	$3,756 63	$10,913 23	$5,290 48	$54,874 48

The work of the parting and refining department requires the services of 27 men, as follows: 1 foreman, 1 plumber, 1 pressman, 1 sweep-washer, 2 engineers, 2 granulators, 2 helpers, 4 reducers, 9 melters, and 4 boiler-men.

The vaults of the establishment usually contain $35,000,000 in gold bricks of various sizes to supply the demand for them. The amount of silver which is usually on hand is about $500,000, of which one-half is in bricks in the vaults, and the other half is in various stages of separation in the parting house.

In conclusion I beg to express my warmest thanks to Mr. A. Mason, the superintendent, and Mr. B. F. Martin, who have most cordially aided me in the preparation of this article.

* Report of the Director of the United States Mint for 1884, p. 47.

THE ELECTRICAL AND MAGNETIC PROPERTIES OF THE IRON CARBURETS.*

DURING the past six or seven years the authors of this paper have been engaged in an extensive series of experiments on the behavior of steel, in regard to the changes produced in its electrical and magnetic properties, by variations in the methods of tempering the metal. These experiments number many hundred, and the results deduced from them are of the greatest interest, and will be briefly described here.

The material used in these experiments is known as "English silver steel." It comes in the form of accurately cylindrical rods of diameter varying from 0.03 to 0.01 centimeter, and of length about 30 centimeters. In hardening different samples of this material, it was found that different degrees of hardness were attained by the different rods when hardened under exactly the same circumstances, but that these different degrees of hardness were separable into several distinct groups, the members of each group behaving similarly. The inference was therefore drawn that the group difference was a difference in chemical composition, and in order to avoid errors due to this difference the inferences were drawn, "as a rule, from rods broken from a single longitudinally homogeneous sample only."

In measuring the resistance of the rods at various temperatures, it was found necessary to work within very narrow limits to prevent any possible change in the rods through annealing in the course of the experiments; the range of temperature adopted was that between the extremes of 10° C. and 35° C. For this range of temperature the formula

$$S_t = S_0(1 + \alpha t)$$

where S_t = specific resistance at t° C., S_0 = specific resistance at 0° C., and α = electrical temperature coefficient for 1° C., is sufficiently accurate for all purposes desired. The values of α to-

* The Electrical and Magnetic Properties of the Iron Carburets, by Carl Barus and Vincent Strouhal. United States Geological Survey. Bulletin No. 14. Washington. 1885.

gether with the specific resistance of steel in different states of hardness, are given in the following table:

	$S \frac{cm}{cm^2}$ 0° C.	α
Glass hard	Microhm 45.7	0.00161
Annealed light yellow	" 28.9	0.00244
Annealed yellow	" 26.3	0.00280
Annealed blue	" 20.5	0.00330
Annealed light blue	" 18.4	0.00360
Soft	" 15.9	0.00423

From whence the law is derived:

The electrical temperature coefficient of steel decreases in proportion as its specific resistance or its degree of hardness increases, at a rate diminishing as we pass from soft to hard steel.

FIG. 1.

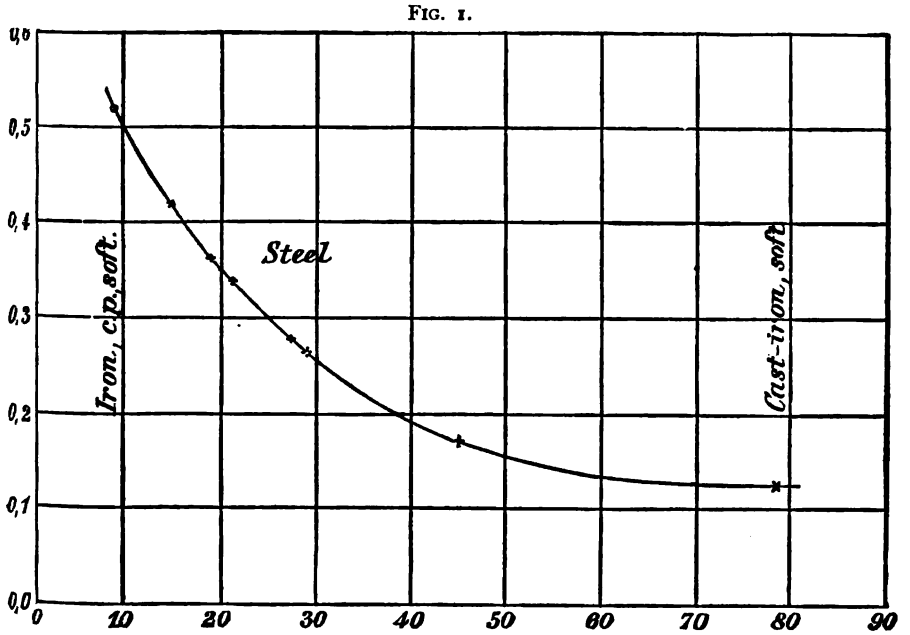


Diagram of the relation between specific electric resistance and temperature coefficient for wrought-iron, for steel, and for cast-iron.

It is also found that these two factors, in the cases of wrought and soft cast iron, fall on the prolongation of the locus of steel.

In attempting to harden steel by heating to successively increas-

ing temperatures, followed by sudden cooling, it is found that but little effect is produced until a certain critical temperature in red is reached, at which hardness suddenly appears; above this temperature hardness increases rapidly at first, and afterwards the rate of increase decreases.

The thermo-electromotive force developed by couples formed of steel and copper was carefully measured in every case, and the results reduced to the equivalent value steel silver. The results show that the thermo-electric force for a difference of temperature of the junction of 1° C. varies with the specific resistance at 0° C., the one increasing as the other decreases, in such a manner that the locus of the curve obtained is a straight line, whose equation is

$$15.176 - y' = (0.4123)\,x,$$

in which y' = thermo-electromotive force for 1° C., and x = specific resistance at 0° C. In this equation, making $x = 0$, we get $y' = 15.176$, that is the thermo-electric force for 1° C. of a couple composed of silver, and of an imaginary steel whose resistance is zero. Making the quantity $15.176 - y' = h$ we have $h = 0.4123\,x$. This quantity h varying directly with x is called the *thermo-electric hardness.* We have previously seen that the mechanical state of hardness varies in the same sense as the specific resistance, so that the thermo-electric hardness furnishes a very convenient method of giving a numerical value to the hardness of steel, the degree of zero hardness corresponding to the imaginary steel whose resistance is also zero. If, instead of adopting this new term, all the measurements of electromotive force were given in terms of y' (silver steel), there would be a change of sign in passing from hard to soft steel, whereas by adopting it all values are positive.

Starting with a sample of glass-hard steel, and exposing it to the annealing effect of steam at 100° for several successive hours, the thermo-electric force (silver steel) was found to vary with the time of exposure, rapidly at first, and then gradually decreasing until it became nearly constant. In the glass-hard state the thermo-electric force being 0.13 microvolts, the various values, after being exposed one hour between each measurement, were 2.75, 3.64, 3.55, 3.77, 3.90, 4.02, showing a gradually decreasing hardness. In a sample annealed in aniline vapor at 185° the results obtained were:

	Glass-hard.	10 min.	20 min. more.	30 min. more.	1 hour more.	1 hour more.
Thermo-electric force microvolts, .	— 0.83	+ 3.59	4.10	4.33	4.61	4.84
Specific resistance microhms, .	39.34	29.25	28.02	27.17	26.32	25.84

For a sample annealed at 330° in melted lead the results were:

	Glass-hard.	1 minute.	30 minutes more.	1 hour more.
Thermo-electric force microvolts, . .	0.08	7.61	7.76	7.80
Specific resistance microhms, . . .	36.04	18.30	18.11	18.02

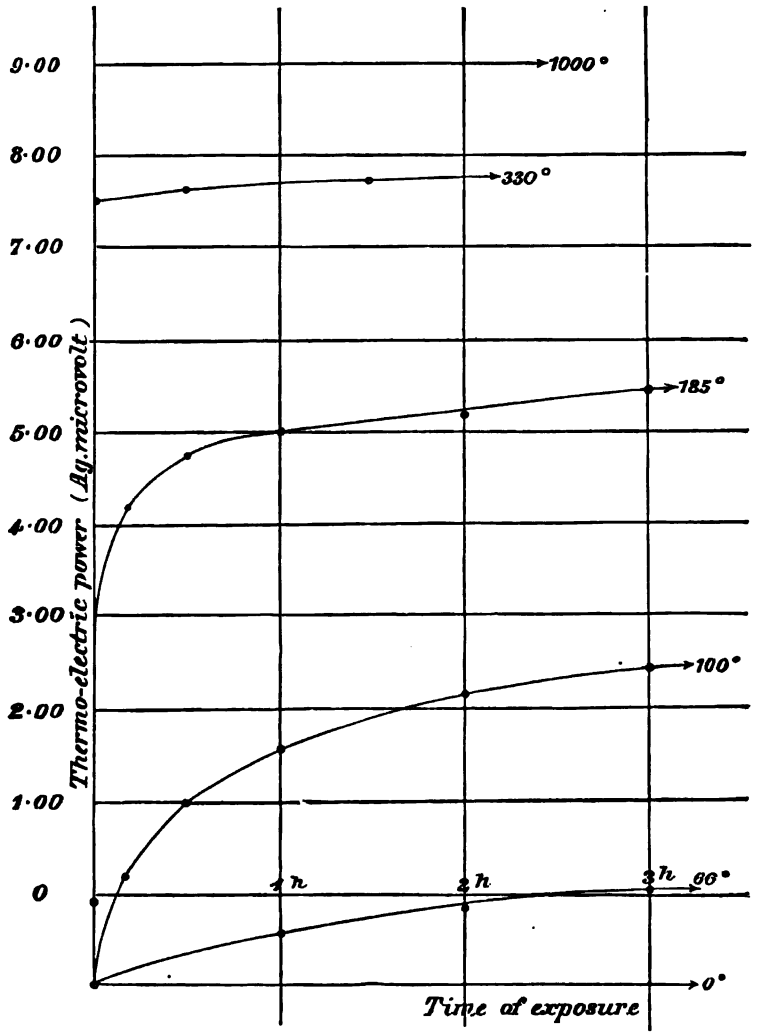
These results show that time is an important element in annealing, more so for lower temperatures than for higher, but that in both cases the action is similar, a very rapid change at first, which continues slowly, decreasing until a fixed state is reached, which remains the same, independently of the time of exposure. After being so annealed a rod will not be further affected by any exposure to a lower temperature than that at which it was heated, while if it be exposed to a higher temperature annealing will recommence, and the final state will be the same as if the rod had not received its preparatory treatment.

The largest observed variation of thermo-electric force obtained by tempering is 12.8 microvolts per degree at 0° C., and the largest observed specific resistance of hard steel is three times that of soft steel.

The phenomena accompanying the process of annealing have given rise to two theories in explanation of the facts observed. The one is the chemical, the other the physical theory. Of these the first assumes that for each distinct state of hardness the carbon has changed its manner of occurrence, that, for instance, with glass-hard steel all the carbon is combined, while on annealing partia. dissociation of the iron carbide takes place setting free so much amorphous carbon, the amount of dissociation depending on the temperature and duration of the annealing process. The physical theory on the other hand assumes that the manner of occurrence of the carbon does not alter during annealing, but that the phenomena are due to molecular rearrangement only. With steel rods hardened by sudden cooling we have the outer shell under a strain of compression, and the internal core under a strain of dilation. The resultant strains are of enormous magnitude, and when the temperature of the metal is raised above that at which it was

chilled, the viscosity of the piece is decreased and an internal flow takes place until a position of equilibrium is reached decreasing the amount of strain on the metal, and thereby influencing its

FIG. 2.



Hard wires annealed continuously at 0°, 66°, 100°, 185°, 330° and 1000° respectively.

thermo-electric, etc., behavior. The chemical theory will probably come into play at that critical temperature in red before referred to, when glass-hardness suddenly appears; below this heat all an-

nealing may be accounted for by the physical theory outlined above. Strong confirmation of this view is afforded by the action of malleable cast iron in the hard and soft states. Here there are comparatively small variations in thermo-electric power and specific resistance for a very great difference in the mechanical hardness. The results obtained on three samples in the soft and chilled hard states are here given :

	Thermo-electric power (microvolts).		Specific resistance (microhms).	
	Soft.	Hard.	Soft.	Hard.
No. 1.....	+ 0.30	— 0.90	23.0	32.7
" 2.....	+ 0.72	+ 0.37	23.6	27.9
" 3.....	+ 1.15	+ 0.73	23.0	26.6

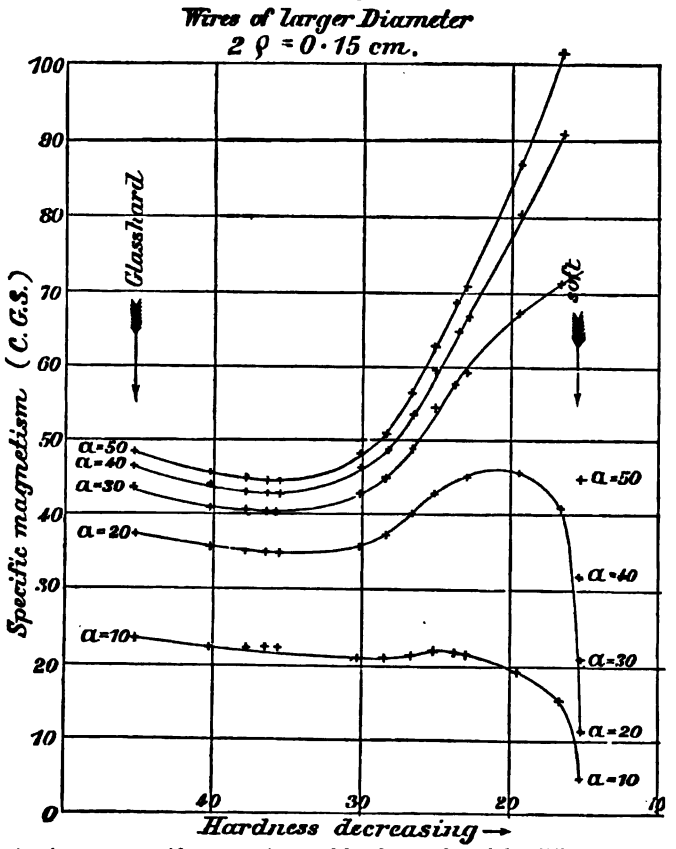
By comparison of these figures with those previously given for steel, the apparently anomalous feature of two materials, so nearly alike chemically, furnishing such decidedly different results, at once causes further inquiry. We come to the conclusion that the mere change of mechanical hardness cannot be the main cause of the change of the electrical properties under discussion, and on inquiring further find that, according to Forquignon : " Le fonte malléable se distingue de l'acier par ses *faible allongements* et sa forte teneur en graphite." The final conclusion then is, that change of internal strain indicated by change of volume which takes place on annealing is the true cause of the varying electrical properties, and that it is " the cause of electrical effects so enormous that such additional effects which any possible change of carburation may involve can be wholly disregarded, and all electrical results interpreted as due solely to variations in the intensity of the said strain."

The red-heat state of iron and steel is remarkable for several characteristic phenomena, the sudden appearance of glass-hardness, very sudden increase of volume, sudden disappearance of magnetism, abnormal thermo-electric behavior, all of which seem to be closely connected in origin.

Steel rods under the influence of the strain caused by magnetization have, as might be expected, a different specific resistance and thermo-electric power than the same steel unmagnetized, but the difference only amounts to a few tenths per cent. (not more than 0.3 per cent. in the most unfavorable case) of the differences

due to tempering the steel. The influence of magnetization may, therefore, be disregarded when comparing steels of different hardness. The electrical effects produced by the magnetic strain are similar in kind to those produced by a tensile strain applied to iron, so long as the iron is not strained beyond its elastic limit. In this connection it is interesting to note the different behavior of

FIG. 3.



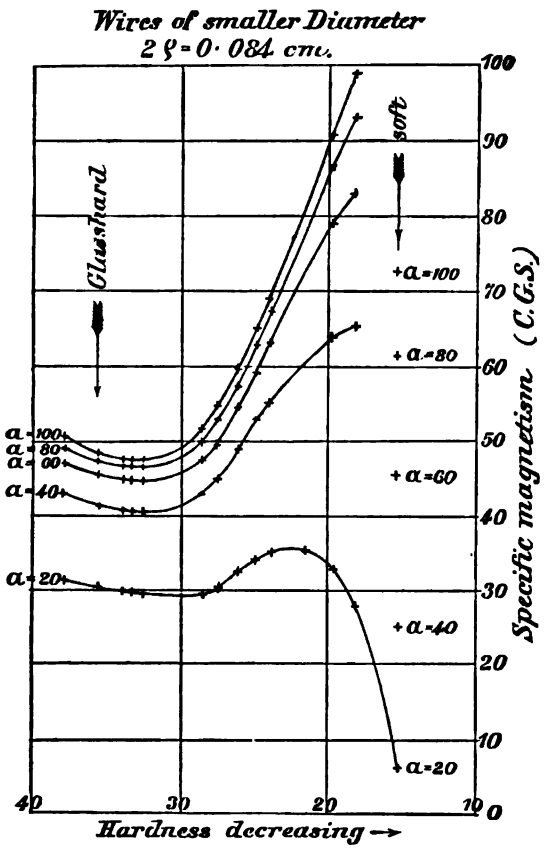
Relation between specific magnetism and hardness of steel for different dimension ratios.

steel hardened by wire drawing and by tempering. The molecular strain in the two cases acts in different directions, in the first axially and in the last radially. In their thermo-electric character they lie on different sides of soft steel, so that the thermo-electric series is :

Sb—hard wire-drawn steel—soft steel—hard-tempered steel—Bi.

The effect of hardness on the magnetizability of steel varies with the dimensions of the rod taken. Other things being equal, soft rods retain more magnetism than hard, except when they are short. For rods in which the ratio of length to diameter is less than 14 or 18 (for the respective diameters 0.08 cm. and 0.15 cm.) the glass-hard state forms the most powerful magnets, when the ratio

FIG. 4.



Relation between specific magnetism and hardness of steel for different dimension ratios.

increases blue annealed rods furnish the maximum magnetizability, and as the ratio continues to increase they are approached by soft rods. For all dimensions of rods, supposed glass-hard and magnetized to saturation at the ordinary temperature, the first effect of

annealing is to diminish the magnetizability to a decided minimum; further annealing then causes an increase in that quality, which, in the case of short rods (ratio 20 and below, depending on diameter), sinks again to a minimum, while with longer rods rises continually towards an enormously developed maximum.

In considering the effect of annealing temperatures on glass-hard saturated magnets we have two causes at work diminishing the amount of residual magnetism. The first and most important is that direct molecular rearrangement due to annealing, which has been already considered. The second is the direct interference of the increase of temperature with the coercitive force of magnetization; this latter effect is, however, of small amount, as previously indicated. By remagnetizing a rod after being thus annealed the loss due to the first cause may be replaced as far as the changed temper of the steel will allow, while that due to the second cause, although replaced, will be lost again when reannealed.

To illustrate this action the following tables will serve. They embody the results of annealing two magnets made from the same rod and as nearly identical as possible. They were annealed in steam, one of them being remagnetized after each exposure, while the other was not. s = specific resistance at 0° C., m = specific magnetism after annealing, m^* = specific magnetism after annealing and remagnetization.

Description of temper.	Magnet No. 11. Length = 10 cm. Weight = 0.417 gram.		Magnet No 12. Length = 10 cm. Weight = 0.418 gram.		
	s.	m.	s.	m	m.*
Glass-hard.....	39.5	62.6	39.9	...	62.5
Annealed 10 minutes at 100° C.	38.7	59.5	39.1	59.7	62.4
20 minutes longer "	37.6	56.0	38.0	58.2	61.9
30 minutes " "	36.7	52.6	37.1	57.5	60.6
1 hour " "	35.7	50.0	36.1	56.5	60.2
2 hours " "	34.8	47.3	35.2	56.1	59.5
3 hours " "	34.1	46.1	34.5	56.4	59.4
4 hours " "	33.5	45.1	33.9	56.5	59.3
5 hours " "	33.1	44.3	33.5	56.3	59.1
6 hours " "	32.8	43.8	33.2	56.5	59.0
22 hours total time of exposure.					

In the case of No. 11 we have in the diminution of m the sum of the diminutions due to both causes of loss of magnetism; in the case of No. 12 the variation in m^* shows the diminution of mag-

netizability due to annealing; we have in the last period of annealing the value $m^* - m$, representing the loss due to the second cause, the direct influence on the coercitive force of magnetization at that state of temper, and under s we have the values of specific resistance changing with the specific magnetism and measuring the degree of hardness.

Both magnets were now remagnetized and retreated as follows:

Time of exposure to 100°.	Specific magnetism.	
	No. 11.	No. 12.
Rods remagnetized	59.0	59.0
10 minutes after	57.3	57.1
20 " more	56.6	56.6
30 " "	55.7	55.9
1 hour "	55.7	55.6
2 " "	55.7	55.6
Rods remagnetized	59.0	58.7
10 minutes after	57.2	57.1
20 " more	56.6	56.4
30 " "	56.0	55.9
1 hour "	56.0	55.9
2 " "	56.0	55.8

We see by these results that magnets so prepared attain, finally, a condition of great permanence in respect to the influence of heat. They may be exposed to any temperature below that at which they were prepared without suffering any permanent loss of magnetism. It is found also that magnets thus prepared will withstand the effects of severe mechanical shocks, and the authors submit the following rules for the construction of permanent magnets:

1. Rods tempered glass-hard are not to be used as essential parts of magnetic instruments.

2. Having tempered a given steel rod in such a way as insures uniformity of glass-hardness throughout its length, expose it for a long time (say 20–30 hours; in case of massive magnets even longer intervals of exposure are preferable) to the annealing effect of steam (100°). The operation may be interrupted as often as desirable. The magnet will then exhibit the maximum of permanent hardness for 100°.

3. Magnetize the rod—whether originally a magnet or not is

quite immaterial—to saturation, and then expose it again for about five hours (in case of massive magnets even larger intervals of exposure are preferable) to the annealing effect of steam (100°). The operation may be interrupted as often as desirable. The magnet will then exhibit both the maximum of permanent magnetization as well as the maximum of permanent hardness corresponding to 100°. Its degree of magnetic retentiveness against the effects of temperature (< 100°), time, and percussion, is probably the highest conveniently attainable.

If we assume that we have a series of iron carbides, ranging from pure iron up to cast iron, with 6 per cent. carbon, other ingre-

FIG. 5.

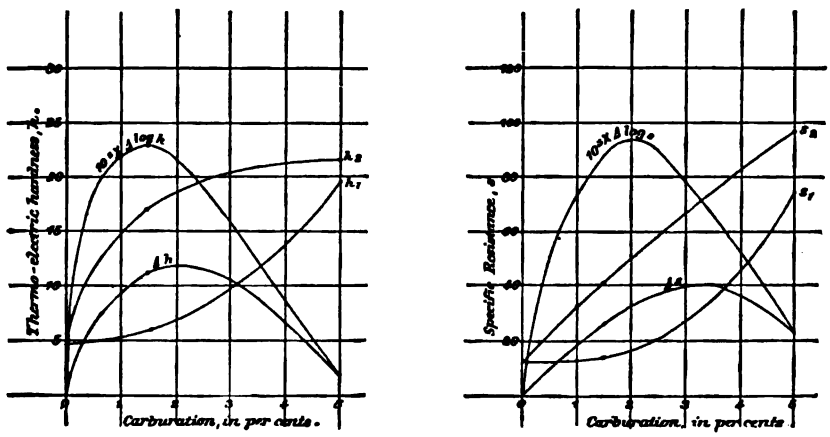


Diagram of the mean variation of the thermo-electric hardness and the specific electric resistance of iron carburets, with their degrees of carburation; 1, Thoroughly annealed state; 2, Suddenly cooled state.

dients remaining unaltered, we will have at a certain range the product known as steel. On comparing the effects of slow and sudden cooling on such a series, by means of the determination of the thermo-electric hardness, and of the specific resistance of the individual members in both states of hardness, we will find, on plotting the curves with carbon per cent. as abscissa, and thermo-electric hardness or specific resistance as ordinate, that in both cases there is an increase of these qualities with increase of carbon. The manner of increase, however, is different. Starting with pure iron where the curves of sudden and slow cooling are coincident, and constructing the curve of sudden cooling, we find that it is

concave towards the axis of abscissas, and on constructing the curve of slow cooling we find that it is convex. These curves will again intersect at some point near 5 or 6 per cent. carbon. If now a fresh curve be constructed, using as ordinates the differences between the logarithms of the pairs of values of thermo-electric hardness, it passes through a very sharp maximum in the region of 1.5 per cent. carbon; if instead of thermo-electric hardness we take specific resistance, a similar maximum is found, about 2 per cent. carbon. The authors propose that this maximum state be taken as defining the normal steel. It is now easy, by determining the maximum and minimum values of thermo-electric hardness of any member of the series, to classify it according to its carbon percentage, other things remaining the same. In the figure we have the two curves, 1 and 2 of the soft and hard states respectively, the curve Δ of the differences between the two sets of values, and the curve $10^3 \Delta$ log. of the differences of the logarithms of the sets of values. †

MEASUREMENT OF LIGHT OBSTRUCTION.

BY JAMES L. GREENLEAF.

FOR ordinary occupations the intensity of direct daylight is often greater than the eye actually needs, and provision is made by nature for shutting off the excess before it can strike the retina. Most occupations, however, which involve the steady use of the eyes are carried on within buildings where the only inlet for daylight is through the windows. Very frequently there is no direct daylight cast upon the work, and it is illuminated solely by the light reflected from the walls and various objects within the room. In such cases the amount or intensity of light thrown upon the work and reflected from it to the eye may be just sufficient to meet the requirements without straining the sight, although the direct sunlight which streams in at the window is too brilliant to be agreeable. Any obstruction of this window light may therefore be a practical injury, and if caused by persons with opposing interests may very justly be made a cause of complaint and claims for damages. This fact has been recognized for years, and many a lawyer's fee has been pocketed in consequence.

Cases are frequently occurring in cities and crowded towns of conflicting interests in light, and the law will sometimes uphold a plaintiff so far as to serve an injunction against the erection of a new building adjoining, or, in certain cases, allow him to build a screen within half an inch of his neighbor's windows, provided it is on his own land.

All modern law appears to the unfortunate army of plaintiffs and defendants, that is privileged to be bitten by it now and then, a vast system of precedents established by our ancestors, and, like them, canonized by reason of their age. The law on "ancient lights," as it is termed, is no exception, and to a great extent, the decisions of the modern court concerning obstructions of light, are governed by what the old Saxons knew of windows long before Queen Elizabeth's day.

There has arisen with the introduction of elevated railways a new class of cases of light obstruction, and of one of these it is proposed to treat. The law holds a railway corporation responsible for injury to previously existing rights the same as an individual, and in nearly all cases where the elevated railways are shown to cause practical injury to property, the courts uphold the plaintiff.

There have been a number of cases of this character tried in New York City, and probably there would be many more except for the fact that a small owner cannot afford litigation with a wealthy corporation. Possibly an organized action of small property owners would be effectual in claiming damages, but experience is against such combinations, and, as a rule, the cases raised are those in which extensive interests are involved.

The injury caused by an elevated railroad is usually a lowering of the marketable value of the real estate beside it, and the amount of this is really the main point at issue. On that point an immense amount of expert testimony is collected *pro* and *con*. Other subjects of testimony are brought in as side matters to convince judge or jury that there is, or is not, practical cause for this alleged depreciation in the value of the property. Of such a nature is the testimony as to the obstruction of light, noise, cinders, and gases, vibrations, etc. This testimony is scientific in its character, and it can be seen that in these cases the scientist or engineer plays second fiddle to the real-estate agent.

The writer had the opportunity of tuning his violin in such a capacity in a case tried recently before one of our New York judges,

and has been requested to lay the methods of his study regarding light obstruction before the readers of the QUARTERLY. The clearest way of doing so is by means of some of the tables used in calculation, and perhaps this fact will be an excuse for offering them, although tables have an established reputation as dry reading matter. Of course it is not claimed that the results are absolutely accurate; this is merely a systematic method of approximation founded on correct principles. As such it is more valuable than guesswork. It is very possibly not the best way to treat the subject, and can properly be criticized.

In estimating the obstruction to the light entering a building, the entire amount of entering light is not calculated. It would be an exceedingly involved operation. For every point taken there would be a new series of angles. It is necessary to fix upon some definite point as a fair average position, and make all the estimates with reference to that. This will give a just comparison of the conditions after and before the obstruction existed.

A point two feet back from the window and five feet above the floor was selected as the point of sight.

The light which reaches this point, provided the sun does not shine directly in, is diffused light. Part may come from the sky, and part from the surfaces of buildings opposite, or from the ground. The obstruction may hide light from the sky or from the buildings opposite in proportions varying with the circumstances. The obstruction itself will give some light.

In order to estimate the amount of light cut off, it is necessary to have a unit. This unit must involve the intensity of the light, and because all the rays converge to the point of sight, it must involve the angular unit of space, if it may be so expressed. The diffused light from the sky was taken as the standard, and the *degree light unit*, as it was termed, is the amount of skylight reaching the point of sight through an angular space of one degree vertical and one degree horizontal; that is, it is the amount of light contained in a square pyramid of rays whose vertex is at the point of sight, and whose opposite sides make an angle of one degree with each other. It will hereafter be designated by the letters D.L.U. The amounts of all lights reaching the point of sight are expressed in terms of this unit. Then comparison is quite simple and the proportion of obstructed light can readily be found.

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The only floors that suffer a practical diminution of light by the elevated railroad are the basement and first floor. This necessitates taking a point of sight in each story, and two parallel sets of calculations. The railroad structure is a permanent obstruction, while passing trains offer obstruction to the light at intervals. The down-train does not, owing to its angular position, very greatly increase the obstruction caused by the structure, and hence only the effect of the up-train is considered. For each point of sight then, there are two sets of calculations to be made: one for the loss of light caused by the structure alone; the other for the total loss when an up-train is passing.

Still another development of the calculations, yet to be explained, is required to allow for the varying intensity of the light in morning and afternoon.

The angles read off from the sections mentioned were entered in a table of which the headings of the columns are given. A table was made for the section at right angles to Beaver St., and another for the section at right angles to Pearl St.

Point of sight.	Total light angle of sky and building.	Total sky-light angle.	Total building light angle.	R. R. structure without trains			R. R. structure with up-train		
				Total angle of shadow.	Skylight shadow angle.	Building light shadow angle.	Total angle of shadow.	Skylight shadow angle.	Building light shadow angle.
1st floor...									
Basement									

Measurement of Light Intensities.

Having obtained the angles needed, the next step is to find the intensities of the lights entering the windows, compared with the skylight as a standard. The lights diffused from the sky, the surfaces of the opposite buildings, the structure of the railroad, etc., all vary in intensity, but in order to know the proportion of total light cut off they must all be measured by the common standard, the D.L.U. Hence the reason for measuring their intensities. This was done by a simple and ingenious method very kindly suggested by Professor Ogden N. Rood.

It consists in rapidly revolving a black disk against the skylight and cutting out segments until the mingling gives a shade similar to that of the surface in question. The proportion of the angle cut out to the entire 360 degrees will give the intensity of the light

from the surface compared with skylight. Let a denote the angle cut from the black disk, which we will assume gives no light. Then a degrees of skylight = 360° of the shade tested. Calling skylight 100 per cent., $x : 100 :: a : 360 \therefore x = \frac{100a}{360}$, which gives the percentage of intensity of the shade, compared with the skylight.

For convenience in making the tests a series of shaded papers were prepared with India ink from white to a dark gray, and these matched with the various surfaces to be experimented upon, and then these papers were used with the revolving disk.

By nearly closing the eyes the element of color is easily eliminated, and a few trials will convince one that the shades can be accurately matched.

In making the experiments the disk should have a high velocity, which is easily produced with a wheel and pulley, and the shaded paper must be held tangent to it against the sky, so that comparison can be readily made.

The disk, which should be about six inches in diameter, can be made of pasteboard, blackened with lampblack. But even this surface will diffuse about 5 per cent. of light. Correction can be made for this by taking 5 per cent. of the angle left of the disk and adding it to the angle a. When the shade experimented with is more intense than the skylight, place the shaded paper back of the disk. Then a degrees of the shade tested = 360° of skylight and $x = \frac{360 \times 100}{a}$.

In the case of the building in Pearl St., the angle with the meridian necessitated tests of shades for the morning and afternoon. Pearl St. runs approximately northeast, and the morning sun shines brightly on the buildings opposite Nos. 140 and 142. About 2.45 P.M. the sun's shadows are parallel with the axis of the street, and subsequently the opposite buildings are in shadow.

The opposite buildings in Beaver St., face approximately south, and the sun can be said to shine directly on them all day. This difference causes an interesting change in the amount of light at different times of the day, as shown by the tables. The amount from Pearl St., and the amount of obstruction are very greatly reduced in the afternoon. The amount both of light and obstruction from Beaver St. remain about the same. By the combination of the lights from Pearl and Beaver Sts., different effects are obtained for different times of day, and also the amount of light entering the windows of No. 140 is seen to differ from that entering No. 142.

All the calculations were made for a clear day, and hence the intensity of light from the following surfaces was required:

Opposite building in Pearl St. with the sun shining brightly upon it (a yellowish-buff color).

The same, slightly in shadow.

The Eagle Fire Insurance Co. building in Beaver St. with the sun shining brightly upon it (light colored brick with considerable white efflorescence, and Belleville brownstone trimmings).

The elevated railroad structure in shadow (a very dark reddish-brown.)

The north to northwestern skylight was taken for the tests.

The percentages of intensity as found by these tests were as follows:

Light from the Pearl St. building in sunlight, . .	127 per cent.
Light from the Pearl St. building in shadow, . .	18 "
Light from the Beaver St. building in sunlight, . .	141 "
Light from elevated railroad in shadow, . . .	6½ "

Allowance must be made for the windows, for which only approximate estimates of the intensity of light can be given. The allowance was made by correcting the percentages given above, for 25 per cent. of window surface with an intensity of 23½ per cent. for the Pearl St. building in sunshine, and 15 per cent. intensity in shadow, making the percentages for the building in sunlight 101 per cent., and in shadow 17½ per cent. And in the case of the Beaver St. building, for 44½ per cent. of window surface with an intensity of 23½ per cent.; making the percentage for the building in sunlight 89 per cent. It will be noticed that the light from the illuminated sides of the buildings was greater than the direct skylight, and the obstruction of the light from them at certain times of day is as serious as the shutting off of the skylight itself.

Calculation of Light Amounts.

After collecting all the data of angles and intensities, the next step is to use them in the calculation of light amounts, and of the percentages of light obstructed.

The D.L.U. has already been defined. It is the amount of the skylight received from a square pyramid of rays of which the point of sight is the apex, and the opposite sides make an angle of one degree with each other.

If the intensity of light from a surface is 89 per cent., then from the same pyramid of rays is obtained 0.89 D.L.U. What follows is a mere matter of calculation of the D.L.U.'s obtained from the various angles of shadow and light found from the sections taken, and their summing up.

The best method of making the calculations is in the form of tables, because the different cases would otherwise get confused.

First, find for each case the total D.L.U.'s without the elevated railroad existing.

Second, find the totals with it in position.

Third, compare the two in the form of percentages and thus get the percentage of light cut off.

Table of Light Amounts under Existing Conditions Omitting Elevated Railroad.

Point of sight.			Total sky-light angle.	Total building light angle.	D.L.U.'s from sky.	D.L.U.'s from building.	Total D.L.U.'s without El. R. R.
Section at right angles to Beaver St.		1st floor....	25.5°	47.0°	25.5	41.7	67.2
		Basement..	27.0°	45.5°	27.0	40.4	67.4
Section at right angles to Pearl Street.	Sun shining on opposite building.	1st floor....	26.0°	52.0°	26.0	52.5	78.5
		Basement..	24.0°	47.0°	24.0	47.4	71.4
	Opposite building in shade.	1st floor....	26.0°	52.0°	26.0	9.0	35.0
		Basement..	24.0°	47.0°	24.0	8.2	32.2

This table gives the total number of D.L.U.'s that reach the point of sight within the space described by revolving a vertical plane through a horizontal angle of one degree. The number of D.L.U.'s are obtained by multiplying the proper angle by the corresponding percentage of intensity of light. For example, the building light angle from Beaver St. is 47°, and the intensity of light from the opposite building is 89 per cent. Hence the number of D.L.U.'s from the building is $47.0 \times 0.89 = 41.7$.

The following table gives the light amounts under the existing circumstances, and also the percentages of loss of light within the vertical section assumed :

Table of Light Amounts Under Existing Conditions.

Section at right angles to Pearl Street.													
Opposite building in shadow.													
R. R. structure with up-train.	1st floor....	26.0°	21.75°	26.00	0.0°	0.00	34.5°	6.00	17.5°	1.11	33.11	35.00	5½ %
	Basement..	24.0°	24.00	0.0°	0.00	25.25°	4.39	21.75°	1.37	29.76	32.20	92½ %	7½ %
R. R. structure without trains.	1st floor....	21.75°	26.00	0.0°	0.00	20.75°	3.61	31.25°	1.97	27.6	35.00	79%	21%
	Basement..	14.25°	14.25	9.75°	0.62	24.0°	4.17	23.0°	1.45	20.49	32.20	63½ %	36½ %
Opposite building in sunshine.													
R. R. structure with up-train.	1st floor....	25.5°	21.75	25.50	4.25°	0.27	20.95	24.23	31.25°	1.97	44.94	78.50	42½ %
	Basement..	18.25°	14.25	18.25	9.75°	0.62	24.0°	24.23	23.0°	1.45	40.55	71.40	57%	43%
R. R. structure without trains.	1st floor....	26.0°	26.00	25.50	0.0°	0.00	34.5°	26.61	17.0°	1.07	53.18	67.20	79%	21%
	Basement..	24.0°	24.00	18.25	0.0°	0.55	25.25°	19.96	22.0°	1.39	48.35	67.40	72%	28½ %
Section at right angles to Beaver Street.													
R. R. structure without trains.	1st floor....	25.5°	25.50	25.50	0.0°	0.00	12.75°	11.31	34.25°	2.16	38.97	67.20	58%	42%
	Basement..	27.0°	27.00	0.0°	0.00	22.5°	19.96	17.30	25.0°	1.58	37.68	67.40	56%	44%
R. R. structure with up-train.	1st floor....	25.5°	25.50	0.0°	0.00	30.0°	26.61	17.30	17.0°	1.07	53.18	67.20	79%	21%
	Basement..	27.0°	27.00	0.0°	0.00	22.5°	19.96	17.30	22.0°	1.39	48.35	67.40	72%	28½ %
Point of sight.		Unobstructed sky angle.	Corresponding D.L.U.'s.	Obstructed sky angle.	Corresponding D.L.U.'s from El. R. R. R.	Unobstructed building angle.	Corresponding D.L.U.'s.	Obstructed building angle.	Corresponding D.L.U.'s from El. R. R. R.	Total D.L.U.'s obtained.	Total D.L.U.'s obtainable if El. R. R. did not exist.	Percentage of D.L.U.'s obtained to what would be had if El. R. R. did not exist.	Percentage of D.L.U.'s cut off to what would be had if El. R. R. did not exist.

It will be noticed in the preceding table that double the number of calculations are made for the Pearl St. section as are made for the section through Beaver St. The reason is that a set is necessary for both the morning and afternoon conditions, whereas the sun shines all day on the opposite buildings in Beaver St. The last column gives for each point of sight the percentage of light cut off within the section taken.

The amount of obstruction in a plane at right angles to each of the streets is now ascertained, and the next, and final step, is to combine these results; to find how much the elevated railroad obstructs the total light which would reach the point of sight, partly from Pearl St., and partly from the Beaver St. opening. Here horizontal angles must be dealt with, and it is obvious from the nature of the case, that the amounts obtained can be only approximate. Still, the same errors that would be made in calculating the amount of obstruction would also be made in estimating the total amount of light, and hence, as proportions only are the final results desired, those results are probably a close approximation to the actual ratios of obstructed to total light.

The method of combining the results already obtained is to consider the two vertical sections revolved through the proper horizontal angles, until on the one hand they meet at the intersection of Pearl and Beaver Sts., and on the other, touch the sides of the window-frame. These angles are the factors which are to be multiplied into the number of D.L.U.'s given in the preceding tables, to find the total number of D.L.U.'s.

Another method of obtaining the final percentages required is to take the percentages of obstruction for each of the sections and combine them according to the proportions that the Beaver St. and Pearl St. angles bear to the entire horizontal angle.

The horizontal angles for the windows of No. 140 vary from those for No. 142, and this necessitates a series of calculations for each division of the building.

The angles as measured from the plan are:

For No. 140, 25° from Beaver St. and 95° from Pearl St.

For No. 142, 50° from Beaver St. and 70° from Pearl St.

Used as indicated, they give the following tables. The required proportions of light cut off are given in the two columns of percentages. The periods of time selected are those for which the light intensities used may be said to hold true. Between 12.30

P.M. and 2 P.M. the proportions would be somewhere intermediate of those given.

Percentage of Light Cut off from No. 140 Pearl Street.

Time of day.		Total D.L.U.'s obtainable if El. R. R. did not exist.	R. R. structure without trains.			R. R. structure with up-train.		
			Total D.L.U.'s obtained.	Total D.L.U.'s lost.	Percentage of D.L.U.'s lost to total obtainable if El. R. R. did not exist.	Total D.L.U.'s obtained.	Total D.L.U.'s lost.	Percentage of D.L.U.'s lost to total obtainable if El. R. R. did not exist.
First floor.	10.30 A. M. to 12.30 P. M.	9138	7213	1925	22%	5244	3894	43%
	2 P. M. to 4.30 P. M.	5005	4475	530	11%	3596	1409	28%
Basement.	10.30 A. M. to 12.30 P. M.	8468	6040	2428	29%	4794	3674	43%
	2 P. M. to 4.30 P. M.	4744	4036	708	15%	2889	1855	39%

Percentage of Light Cut off from No. 142 Pearl Street.

Time of day.		Total D.L.U.'s obtainable if El. R. R. did not exist.	R. R. structure without trains.			R. R. structure with up-train.		
			Total D.L.U.'s obtained.	Total D.L.U.'s lost.	Percentage of D.L.U.'s lost to total obtainable if El. R. R. did not exist.	Total D.L.U.'s obtained.	Total D.L.U.'s lost.	Percentage of D.L.U.'s lost to total obtainable if El. R. R. did not exist.
First floor.	10.30 A. M. to 12.30 P. M.	8855	6994	1861	21%	5094	3761	42%
	2 P. M. to 4.30 P. M.	5810	4977	833	14%	3881	1929	33%
Basement.	10.30 A. M. to 12.30 P. M.	8368	5978	2390	29%	4723	3645	44%
	2 P. M. to 4.30 P. M.	5624	4501	1123	20%	3318	2306	41%

Having derived the last two tables, the object of this paper is accomplished, viz.: to explain a method of finding the amount of light cut off by an obstruction. The use to be made of the results depends on circumstances and is foreign to our purpose, but never-

theless a brief comparison of these results of calculation may not be uninteresting.

The play of the angles, and the different intensities of light, varying with time of day, one upon the other, can be traced back until the study becomes quite complicated. One or two facts, however, stand out more conspicuously from the tables.

It has already been noticed that the light diffused from the opposite buildings is practically just as valuable, even after making a liberal discount for windows, as the direct skylight. This is contrary to the impressions of most persons who have given the subject no special attention. The general opinion is that the skylight is the light to be defended. This impression is true enough, except for times when the sun's rays are illuminating the opposite buildings.

The number of D.L.U.'s from the Pearl St. building in shadow is less than 20 per cent. of those given with the sun shining upon it. In cloudy weather the intensity of the skylight is diminished, and all the buildings may be considered in shadow. Then, as in the afternoon, the skylight is more intense than the building light.

The influence of the building light is shown in the fact that after combining the building light and skylight in the Pearl St. section, the total number of D.L.U.'s for afternoon is less than half that for morning, although the amount from the sky remains practically the same. And this in spite of the fact that the sun is moving so as to shine more toward the east side of Pearl St. In short, the opposite building acts like a great reflector, or more properly, diffuser.

Another fact seen from the tables is that about 25 per cent. of the morning light is cut off by the structure alone, and when an up-train passes, nearly one-half of the light is obscured.

It would naturally be thought that this amount of obstruction would keep the offices in a continual twilight, and yet the effect to the casual observer is not very apparent. There is, to be sure, a shade cast upon the windows, and when a train passes, a very decided flickering of the light, but the effect upon the eye is not such as to make one feel that 43 per cent. of the light has been cut off. It is an illustration that the eye cannot measure at all accurately even a very extensive change in intensity of light. This change, nevertheless, may be a decided detriment to the building, as has already been pointed out in the first sentences of this paper.

A means is afforded of roughly comparing the values of locations on the east side of Pearl St. and on the Beaver St. opening as regards light privileges. Looking at the table of "light amounts under existing conditions, omitting the elevated railroad," it is seen that during the morning of a bright day the amount of light is about the same in both cases, but during the afternoon a building facing the Beaver St. opening has twice as much light for its lower floors as one on Pearl St. only. The comparison would not hold good, of course, for another arrangement of streets, or for buildings painted another color.

During cloudy weather, a building on the Beaver St. opening would have a decided advantage all day long. Not necessarily because the angle of skylight is any greater,—in the present case the sections show it to be nearly the same—but because there is more opportunity for skylight to strike the opposite building, the Eagle Fire Insurance Co. building, giving it greater power as a source of diffused light.

The influence of the Beaver St. opening is shown in the different effects upon Nos. 140 and 142. The latter division of the building is nearer the opening, and has a greater horizontal angle in Beaver St. than No. 140. The total light obtainable for No. 140 in the afternoon is 55 per cent. of that in the morning. For No. 142 it is 66 per cent.

The same point is shown by the last two tables in the percentages of light lost. In the morning the loss averages 25 per cent. and 43 per cent. for the structure without and with trains respectively, for both divisions of the building. In the afternoon it averages 13 per cent. and 33 per cent. for No. 140, and for No. 142, 17 per cent. and 37 per cent., an increase of 4 per cent. The difference is due to the difference in intensity of building light from the two streets in the afternoon, and to No. 142 having a greater horizontal angle in Beaver St. than No. 140.

BASIC OPEN-HEARTH STEEL PROCESS.

BY T. EGLESTON, PH.D.

THE manufacture of basic or Thomas-Gilchrist steel in the open-hearth furnace is so little known in this country, and possesses so many advantages, that it seems a matter of interest to present a careful review on this subject.* The process has been used to a considerable extent in France and Russia; it is being introduced in England, and occupies a position altogether different from that of any other of the steel processes. Nearly 1000 tons per week of this steel is made in various works on the continent of Europe. Experiments have been made in England where a few furnaces have been built and works are now in course of construction or in contemplation. It is very applicable to the conditions of some places in this country. As there is at the present time the greatest possible interest in all processes which can produce large quantities of high-grade steel of uniform quality, I have thought it worth while to make a résumé of what has been published on this subject, and to add to it such notes as I have been permitted to take while studying the process in the various works which I have been allowed to visit, and such others as I have been able to collect from correspondents and others using the process.

Great apprehension was felt at first with regard to the successful application of basic methods to the open-hearth process, so that the furnaces were all made small, too small to be commercially successful. These fears have proved to be unfounded, so that the furnaces are now being built of the same capacity as other open-hearth furnaces.

Experience has shown that it is necessary in the basic Bessemer process in order to make good steel to have from one to two and a half per cent. of phosphorus in the material used. The moment three per cent. is reached, or less than one per cent., the loss in iron necessary to insure the removal of the phosphorus becomes so great on the one hand, and the difficulties of the process so much increased on the other, that this method loses most of its advantages. The quantity of silicon also is limited to from 0.5 to 1.8 per cent., with the general experience that the less of it there

* Basic Refractory Materials, Trans. Inst. Min. Engs., vol. xiv. Basic Bessemer Process, Proc. Am. Soc. Mech. Engs., vol. vii.

is the better, in order to obtain which the manganese is required to be from 0.5 to 2.5 per cent. Such irons are not always easy to get or to make by mixture, though they can be had. There is a very large amount of material which might be converted into steel, which, however, may contain more than 1.8 per cent. of silicon and less than 1 or more than 3 per cent. of phosphorus, and a great deal of wrought-iron scrap, that contains as low as two or three-tenths of 1 per cent. of it, which is altogether unsuitable for either the basic or acid Bessemer or the ordinary open-hearth process. It can, however, be used in the basic open-hearth process with very great advantage.

The basic substances which are used for the refractory materials in the open-hearth process are shrunk dolomite for the lining, magnesia, magnesite, bauxite, and chrome iron for the points of contact between the basic materials of the hearth and the acid bricks of the roof. As the dolomite requires to be burned at a very high temperature, the ordinary shrunk dolomite lining of the cupolas was soon abandoned in Russia, where it is now* burned in iron-jacketed kilns 3 meters high by .75 in diameter, the lining of which is made of large blocks of chrome iron laid up with a mortar consisting of two parts of powdered chrome iron and one of lime. To light the furnace it is filled with wood to the depth of 1.25 m., and then with 60 cm. of small coke, after which 560 kilos of dolomite and 260 to 300 of coke are charged. The charge is drawn twice in every twenty-four hours, giving 320 kilos of calcined dolomite, or 51 per cent. of the weight of the raw material. The calcined dolomite is sorted into three sizes, the first of which is hard burned and is very dense, and is used with the tar mixtures; the second is intermediate, and is used with the first; the third is light and porous, and amounts to about 25 per cent. of the charge, and is used exclusively for repairs.

The outside of the furnace, which is used for the basic process as generally constructed, does not differ essentially from that of the ordinary acid open-hearth furnace. The hearth is cooled by air-channels, which pass under the inclined plates which form the bottom, and which connect with six or more iron chimneys, which rise above the top of the furnace. It has, however, been proposed recently to make important changes in this respect.† The

* Stahl und Eisen, vol. ii., p. 603.

† Journ. Iron and Steel Inst., part ii., 1884, p. 453.

regenerators instead of being placed under the furnace are placed outside of it, as is usually done in Europe when they are built in wet ground. Both the furnaces and regenerators are made round, and with movable roofs built in iron rings, and made tight by sand joints about three inches deep, so that they can be easily detached for repair to the interior, or others put in their place if repair is to be made to the roof. The under side of the hearth is exposed to the air and is in this way subjected to constant radiation, and is found to wear much less rapidly, while its round shape is specially adapted to good working. The regenerators are also built in iron mantels, and as their roof is movable also, are accessible at all times for cleaning or repairs. The interior of the furnace, however, requires certain modifications which are necessary on account of the high quality and price of the refractory materials, and the difficulty also produced by the chemical changes which are likely to take place between a roof which is necessarily acid, and a hearth which must be basic. For a long time it was thought that it would be almost impossible to solve this problem in a commercial way; the study, however, which it has been necessary to give to the basic refractory materials, has shown that this problem can be solved, and that in the near future it is quite likely that materials which have commanded so high a price as to be altogether unsuitable for the process, may be had in the natural state, or made artificially on a large scale, and so cheap as to be within the reach of every one.

It is necessary in constructing a furnace that the roof should be separated from the hearth, both on account of the chemical reactions which are likely to take place between them, and also to facilitate repairs to the junction of the materials which form the hearth and side walls with that of the roof, as these have to be very frequently made. In old furnaces or furnaces constructed after the old pattern, in order to do this, at the spring of the arch forming the roof, plates of cast iron are bolted to the iron plates which form the outside of the furnace in such a way as to make the roof entirely independent. They do not, however, project so far into the interior as to be sensibly affected by the heat of the furnace. On these plates the arch, which is always made in Dinas brick, is sprung. This arch is made as is usual in most of the works in England, where it simply rests on the ramming of chrome iron or magnesite above the shrunk dolomite ramming, but in Russia one arch is made to spring from each side of the gas and air ports towards

the centre of the hearth for two-fifths of the distance, and another one over the remaining three-fifths. The entire roof is thus made of three distinct arches, so that each part of the roof can be repaired independently. Roofs have been built both of shrunk dolomite bricks and of magnesite, but they have not stood as well as the acid roofs made of the best quality of silica bricks, when the junction between the basic and acid material has been properly made. As there is danger of the ports becoming damaged, some special construction is usually devised for them. In Russia they are carried up separately through the roof and wedged to it. In order to overcome the same difficulty it has been proposed at Farnley* to construct both the air and gas ports outside of the furnace, and to make the roof with a very low arch. In most of the English and other works, however, the gas and air ports are built of silica bricks, as is usual in the ordinary open-hearth furnaces. The analysis of the bricks used at Brymbo, which are of exceptionally good quality, is given below:

Si_2O,	93·75
CaO,	3·19
MgO,	trace.
Fe_2O_3,	1·05
Al_2O_3,	1·45
	99·44

A large part of the difficulty of starting the process was owing to the fact, that in almost every instance where the process has been tried, there had been no previous experience with basic material, and also that the works had old furnaces which they adapted to the new process. The furnaces in Russia, and in most of the English works, however, were built exclusively for the purpose, those in Russia having been built with the three arches, while those in England are constructed with a more or less flat single arch. Either way seems to be very advantageous, as each gives excellent results. None but acid furnaces have as yet been built with movable roofs, but this construction would be a very advantageous one, for where the roof is movable, it is entirely independent of the rest of the furnace, and only rests on it while in work, but it must be weighted down with cast-iron to prevent its rising.

* Proc. Soc. Civ. Engs., London, vol. lxxvii., 1883–84, part iii.

All the joints must for this purpose be carefully prepared and closed with sand so as not to allow any inlet of air at that point.

In the Russian works the hearth is made by building up first the gas and air ports with silica brick, and on these the bottom is made with burned dolomite. It is necessary that the dolomite should be burned at a very high temperature, so that it will not change its form. It is ground up to the size of a pea and mixed on a heated iron plate with from 8 to 10 per cent. of tar. This mixture is quite dry, but is all that is needed if the dolomite has been well burned and not ground too fine. Badly burned or too finely ground shrunk dolomite requires more tar, is more difficult to use, and gives less favorable results. The mixture so made is stamped in with hot stampers, adding but little at a time, to a thickness of 38 cm. The side walls are then made by putting up a temporary wooden wall inclined towards the sides, but parallel to them, and distant from the side wall 46 cm. below and 29 cm. above. This is filled in in the same way as the hearth, but as many lumps of hard burned dolomite the size of a walnut as the material will conveniently hold are stamped into it. The work is carried on in this way up to the spring of the arch, which is about 0.45 m. above any possible slag line. All round the furnace on the sides, the junction between the acid and basic material is made by ramming about 0.10 m. of the mixture of tar with either chrome iron or magnesite, but at the ends at the gas and air ports where the heat is greater, and where there is a constant action of fine lime carried by the blast, the separation is made by leaving a V-shaped space in the basic lining of about 15 cm., into which a mixture of ground chrome iron-ore and tar is carefully rammed. As the chrome iron is practically infusible, this separation, which causes no difficulty, prevents any action at the most critical point, which on the sides is only slight. The chrome iron must, however, be quite pure. It has often failed in England on account of the impurities, which were allowed to remain in it. The following is the analysis of that used at Brymbo:

Cr_2O_3,	39.52
Al_2O_3,	4.18
FeO,	19.68
MgO,	25.01
SiO_2,	0.03
Loss,	11.
	99.42

This has been made into a strong brick at these works by mixing it with water and burning it, but it has not been used in the furnace.

To make the hearth, two men stamp the bottom, two men bring the mixture, two men make and grind it. It takes four days to stamp the bottom, and three days to put in the acid roof and ports, or seven days to prepare the furnace for firing. So constructed, the furnaces last from six to seven months with only very slight repairs.

The charging-door holes are arranged so that repairs to the lower part of them, where the greatest chemical action takes place, can be easily made. There are always two of them. Under their sill over which the slag is drawn, the thickness of basic lining is made at least 0.25 m. The door itself is made with a wrought-iron frame filled with silica bricks, which last from ten to fifteen days. The tap-hole is made directly opposite to the charging-door. This is made in a frame bolted to the outside plate of the furnace which can be easily removed. It is provided with a repair-piece fastened to it, in which the tap-hole is made. Sometimes two holes are cast in this plate, but generally there is only one. The tap-holes are formed by inserting wooden plugs through the cast-iron plates and ramming round them. They are left in place, and are burned out when the furnace is fired. The tap-hole should slant sufficiently to make the metal run easily, and should completely drain the furnace. The spout is attached by lugs cast on each side of the iron repair-piece so that it can be easily removed for repairs. It is packed with sand. In Russia the outer end of the spout is closed, the tapping being done through holes in the bottom. Generally there is only a single opening in the iron mantle on the tapping side of the furnace, and this is directly over the tap-hole. In some works there are two made just opposite to the working doors. These openings are bricked up while the furnace is in work. They are made to facilitate repairs.

When the roof of the furnace is made movable the hearth and roof are made with but little difficulty; when it is not the silica roof is removed to stamp in the hearth. The removal of the roof not only allows of repairs being easily made, but of stamping in the hearth with much greater certainty and celerity than can be done when the roof is permanent. Repairs are also greatly facilitated, as there is no necessity of cooling down the furnace to repair the roof, which may be removed to any convenient place. Repairs

are also much more easily made to a round hearth than to a rectangular one. When the generators are above-ground the ports are also easily accessible for repairs.

At Dombrowa, both the hearth and sides of the furnace are made by ramming calcined magnesite, from Styria, just as the shrunk dolomite was previously treated. The analysis of this magnesite is given below:

Magnesia carbonate,	90.30 to 97.32
Calcium carbonate,	0.05 to 0.61
Clay,	1.40
Iron oxide,	4.49
Insoluble,	3.73
	99.97

The roof has also been successfully made of bricks of the same material, and has lasted six to seven months with but little repair. The high cost of the magnesite has, however, prevented its general use.

Magnesia is now being made from sea-water so cheaply and to so large an extent, that it is altogether probable that in the near future it will be used for the hearth, and possibly for the roof, as it is the best and most durable refractory material,* when made with an hydraulic press and properly burned.

When the hearth is completed it is entirely covered over with rails. The sides are protected by a boiler plate against which rail ends are propped to prevent the action of the air upon the bricks. When the furnace is new it is usually fired for twenty-four hours with wood and then heated with gas. The shrunk dolomite mass is thus brought up to a temperature at which it will almost become soft before the tar is entirely burned away. The furnace is then fired and the temperature brought up slowly to a little below a white welding heat. When there would be danger that the iron would weld together, it is removed. The bed has then become so hard that it will resist the blow of a pick. In Russia it takes usually one to one and a half days with wood, and two with gas, to bring the furnace up to the point of driving all the tar out of the lining. It will be from seventy-two to eighty-four hours from the start before the furnace is ready for the charge. In England, on

* Basic Bessemer Process, Trans. Am. Inst. Min. Engs., vol. xiv.

account of the care which is necessary with the acid bricks, it generally takes a week to fire the furnace.

At Farnley, in England, the construction of the hearth was slightly different. The center of the bottom of the furnace was first put in with concrete pounded up through the center of the bottom and on the sides. Over this the tar bricks were piled in such a way as to make interstices and steps, on which the upper lining could be rammed. This was brought up to within 15 or 16 cm. of the ports and of the roof support. On top of this, between the basic lining and the acid roof, magnesia mixed with tar, and above it bauxite mixed with tar, was rammed, so that the bauxite was about three times as thick as the magnesia, and formed the junction between the magnesia and the acid roof. Exactly the same thing was done where the basic lining comes in contact with the silica lining of the air and gas ports. When nothing is used between the acid and basic linings, as formerly at Dombrowa, the action between the two is prevented to a great extent by frequently changing the air and gas.*

At Farnley when the furnace was so far finished it was lighted, and in seventy-two hours the bottom of the furnace upon which the charge was to rest was put in. This was done with freshly burned dolomite, mixed with 10 per cent. of ground fire-brick, so that there was just enough silica with the highest heat of the furnace to form an adherent mixture. After the hearth had been smoothed down to its proper shape it was ready for the charge. The blocks which are used for the tap-holes were burned out during the heating, and they were filled with dry lime. In some works from 0.5 m. to 0.8 m. of the shrunk dolomite, without tar, is thrown against the tap-hole, on the inside of the furnace, from the working door. Then from the outside of the tap-hole sand is put in several centimeters in depth. The whole of the rest of the tap-hole is then filled with anthracite coal, in small pieces, which is prevented from burning by clay rammed in at the end of the tap-hole next the outside of the furnace. When closed in this way the tap-hole gives no trouble in firing, and can be opened in a few minutes without difficulty at the time of casting. In Russia the tap-holes are filled up with lime or coke dust and tar.

The method of conducting the operation, though essentially

* Berg. und Hütt. Zeit., 1885, p. 225.

the same, differs somewhat in its details. In order to get a sufficient quantity of basic slag, about six per cent. of hot limestone is spread over the hearth. Pig is then charged on the center of the bed of the hearth, and covered with as much scrap as the furnace will hold. It will take from three to four hours to melt these together. The remaining scrap is then added, from 300 to 320 kilos at a time. This must be done carefully so as not to chill the bath. On account of the difficulty of making partial charges in a hot furnace, some of the managers prefer to add the whole of the charge at one time, and thus avoid the reduction of temperature from the addition of cold scrap, or to have the whole of the scrap heated to a red heat or even melted before it is put in. The slag which first forms is very fluid and is tapped out from under the working door as soon as it forms, as this is only very slightly higher than the surface of the bath. The slag is not allowed at any time to accumulate in the furnace. As it contains 80 per cent. of the phosphoric acid in the charge, it is kept in contact with the bath for the shortest possible time. When there is danger that there will not be enough to cover the bath, a dam of basic material is made at the working door, or the slag would run over the sill almost continuously. Occasionally the slag becomes too pasty, when a little silicious material is added in the shape of silicious limestone to thin it down. When the whole charge is melted it is crassed, and when entirely clean, rabbled, and tests taken. The pieces taken to be tested are of a conical form, and are hammered down to a disk of 5 mm. thick, cooled in water and bent double. If the charge is working properly the fracture will be fibrous without any traces of crystals. If there are crystals, in Russia, 60 kilos. of bricks, made of a mixture of slaked lime and roll scale, are added. Towards the close of the process this mixture is exclusively used in place of the lime. After the removal of the slag the bath is tested again. If there are still any crystals in the test piece the same thing is repeated until no crystals are shown. At this time it will be necessary to bend the assay several times backwards and forwards in order to break it. Pig iron is then added and another test taken, and then the red-hot spiegel. The final test, after the addition of the spiegel, is made by allowing the sample to cool to dull red heat and then quenching in water. This must stand 6 or 7 blows of a 12-kilo. sledge before breaking. If it breaks before the sixth blow the metal is too hard, but if it re-

quires more than seven, more spiegel is added in the furnace. At this point the ferro is added and rabbled through the charge. Sometimes a pole of green wood is inserted into the bath for a few moments to ensure its being entirely homogeneous. If the charge has been properly made, the metal will flow quietly when tapped. Such metal as has a tendency to rise in the moulds must either be cooled with water or compressed to prevent it. It is usual also to take tests during the casting to ascertain the exact quality of each ingot. Two of these are bent so that they will break. Two others are bent at right angles and beaten back on an anvil, being turned over at every stroke, and must stand 25 to 30 blows before they break. The charge takes nine hours and a half, and after it is withdrawn two hours and a half are required for making the repairs, so that only two heats a day can be made.

In England* the limestone is added just as it is in Russia. Its weight is from one-seventh to one-tenth of the weight of the charge. On this the charge of pig is placed, and over it the steel crop ends or raw scrap. The charge is introduced cold, and if possible the whole charge is put in at one time. At the works of the Brymbo Iron and Steel Company the pig contains:

Carbon,	3 per cent.
Phosphorus,	3 " "
Silicon,	0.4 " "
Manganese,	1 to 2 " "
Sulphur,	0.03 " "

The iron is made very low in silicon by inclining the tuyeres of the blast furnace so that the iron is fined and freed from silicon in the furnace itself.

Eighty per cent. of this pig, with 20 per cent. of scrap, are used. The charge is thus made up of pig 60 per cent., scrap 15 per cent., ore 10 per cent., limestone 15 per cent. by weight. The iron ore used is calcined black band. The limestone is broken to about the size of a walnut. Taking the charge by weight it is:

Pig,	7 tons 10 cwt.
Steel and wrought scrap,	2 tons 10 "
Limestone,	2 tons.
Iron ore,	1 ton 17 cwt.

* Proc. Inst. Civ. Engs., vol. lxxvii., 1883–4, part iii.

The whole charge of scrap and pig is weighed out, and placed in front of the furnace. The limestone and ore were also weighed out formerly, but it has been found that to do the work properly, sometimes a little more, and sometimes a little less, is required, so that now the men are allowed to have what they need. The calcined ore and the limestone are placed near to the furnace, where they are most easily accessible. Charging the ten tons of pig and scrap usually takes an hour. The melting takes three to four hours. The working of the charge takes seven to ten hours, so that the whole operation will last from twelve to fifteen hours. The slag is allowed to accumulate in the furnace for the first two hours, after that it is tapped regularly every hour and a half. The proportion of phosphorus which it contains varies very much in the different stages of the process, the highest being shortly after melting. The average composition of the slag at Brymbo then is:

SiO_2,	10.00
P_2O_5,	23.05
CaO,	45.37
MgO,	9.12
Fe_2O_3,	2.29
FeO,	3.29
Al_2O_3,	1.05
MnO	4.20
	98.37

By carrying on the operation long enough the whole of the phosphorus might be eliminated, but it would be tedious and expensive, and would not pay for the labor and trouble. As soon as the pig is melted the charge will have contracted sufficiently to allow the slag to float, which is then removed at once. The first slag is generally very fluid, and as it contains the larger part of the silicon it is removed as quickly as possible. It is essential that these first slags should be withdrawn as they form, both to remove the phosphorus and silicon they contain as well as to prevent any attack on the sides of the furnace. If the slag is very fluid lime is added to dry it up, and the slag is then again removed, sufficient only being left to protect the iron in the ladle. The work of the furnace consists in constantly watching it to see that it is supplied with calcined ore and limestone, two or three shovelfuls of each being thrown in as soon as the men see that the previous additions have been absorbed

in the slag. This will be every three to five minutes. The limestone is broken up to the size of a hickory nut, but the ore is in pieces two or three times the size of the fist. It must not, however, be of too large a size. The pressure is so regulated, that the gas tends rather to come out of the furnace doors than the air to enter. When the slag is to be tapped, the basic, which is from 0.10 m. to 0.15 m. higher than the acid sill of the working door, is cut down with a bar about 3 m. long, manipulated with a sledge, and cleaned out with a hook on the end of an iron rod 4 m. to 5 m. long. The slag flows quietly over an incline bank of sand, in front of the slag door, into a sand puddle about 1 m. square. The slag cools quickly, so that the crust has to be broken every few minutes to allow it to rise. It is cooled with water and taken away. The sand barriers are then shoveled up against the sand incline to be used over again. The cutting down of the basic material allows the slag to flow under the working door, which is kept down all the time the slag flows. The doors of the furnace are opened as little as possible, and are kept open only a few seconds at a time. Just as soon as the slag ceases to flow, the channel is filled up with fine shrunk dolomite about the size of a hazelnut. The slag is then cut away from the sill, which is made of acid bricks, which are then covered with a very dry mixture of shrunk dolomite and tar. Occasionally the slag breaks through, when the repairs are made in the same way. The doors of the furnace lift in opposite directions. Both for convenience of working, and to avoid the heat, the slag pools are made at opposite ends of adjoining furnaces. In throwing in the fluxes, one man throws in the limestone, and the other the ore. The door of the furnace is lifted for them by a boy, and is closed as soon as a single shovelful is thrown in, and is raised again for the next one, and so on. The slag could easily be tapped into an iron car and be run off, so as to be out of the way, and thus avoid the labor of handling it so often. It has now to be handled two or three times before it is eventually disposed of. These slags might be treated,* but they are always thrown away. This is all the work the men have to do, but it is difficult on account of the heat, and requires considerable judgment. The gas and air valves are changed every twenty minutes to every half an hour, depending on the heat of the furnace. The gas, when the

* Basic Bessemer Process, Trans. Am. Soc. Mech. Eng., vol. vii.

furnace is working well, burns with a flame tipped with bluish-green. When the whole charge is entirely quiet it is rabbled and a sample taken. The first sample is made into an ingot 98 cm. in diameter, and about 5 or 6 cm. thick. It is hammered in its own heat, quenched with water, and broken into two pieces. Any long streaks in the fracture of a dark color, or crystals, are indications of the presence of phosphorus. If the sample is not right the slag is removed, it being too pasty to run; it is drawn out by the center door, which is 15 cm. lower than the charging door. About five per cent. of hematite and more limestone is charged near the bridges. This causes a violent ebullition in the bath, which lasts for about half an hour, when another sample is taken. The men are able to tell with great nicety from the look of the fracture how much ore and limestone are required to remove the last traces of phosphorus. In a quarter to half an hour any slag which is in excess over that required to afterwards protect the metal in the ladle, is taken off and the spiegel or ferro is then added. In England it is generally considered desirable to add the ferro in the ladle, as the reactions which take place are very violent, and it cannot very well be done in the furnace. It makes the steel more uniform, however, to add it in the furnace, but there always is danger of the phosphorus going back into the metal. Some slag is always left to cover the metal in the ladle to avoid the chilling of the steel while casting and thus save a considerable quantity of it. Some is always left in the bottom of the furnace, and this and any other is always raked out at once, especially around the tap-hole, for if this is not done it is difficult to clear it after it gets a little cool.

At Dombrowa about 100 kilos. of limestone, previously heated, is first spread over the hearth. Iron ore and cast iron are then charged on the middle, and on the inclined sides near the ports the heavy pieces of wrought iron, with steel scrap. All of these materials except the ore are heated to redness before being charged. The bath is skimmed from time to time, lime being added according as the test shows crystals or hard spots in the test pieces. The spiegel or ferro is added when the charge is ready for it. From 8 to 11 tons are produced in 7 to 8 hours.*

Generally the contents of the furnace is tapped into a ladle and

* Berg. und Hütt. Zeit., 1885, p. 225.

cast from that into the ingot moulds; five to six hours before casting, the red-hot slag is piled up in the spout to dry it, and a fire is made in the ladle to heat it up. In some of the works, as the metal contains so little carbon that it is pasty and difficult to keep in a fluid condition, the ladles have two goose-necks, so that they can dispose of the metal very quickly, but generally the ladles have only one. The tap-hole is cleaned up to the sand packing, and the ladle brought up under the spout. The ladle is either on a carriage, which is actuated by steam or by hand, by means of a wheel or worm, or a hydraulic locomotive crane is used. Generally there will be three ladles for four furnaces, one of them being kept in reserve. There should be only just enough slag left in the furnace to cover the metal in the ladle when the furnace is tapped. As soon as all the metal is in the ladle, about 90 pounds of ferro, broken to pieces about 0.03 m. cube, are thrown into it. This ferro contains from 79.76 to 82 of manganese, and from 0.10 to 0.14 of phosphorus. The metal is allowed to rest for an instant, and is then cast. As soon as the ferro is added, a brown smoke flares up from the surface of the metal. It has been found that when the ferro is added in the furnace, some of the phosphorus from the slag goes back into the metal. If it should prove that there was more than enough slag to cover the metal it is turned into the pit. The slag in the ladle soon cools, and forms a cover over the melted metal, which keeps it from being chilled. The casting is made both top and bottom. Sometimes the ordinary funnel-shaped spout is used in bottom casting to convey the metal to the runners, and sometimes a large ingot mould. When a very large quantity of metal, however, is run through an ingot mould, it is very apt to crack. Frequently as many as eight or nine ingots are cast together in this way. The runners for the bottom casting must be made quite strong, as they sometimes break and let the metal flow into the pit. The ladles last, with patching, from three to four weeks. They are easily patched. The weight of ingots obtained, when there is no accident, is 5 per cent. less than the weight of the pig and scrap of the charge.

In Russia the casting is done into ingot moulds placed on a revolving table, no ladle being used. When one mould is filled the tap-hole is closed by a bar, and the table revolved until the next mould comes under it, and so on. When the steel is all out the slag is allowed to drain for a few moments, but no attempt is made

to get it all out. The spout is removed; the tap-hole is then closed with coke dust and tar, or with shrunk dolomite, sand, anthracite, and clay. The spout is then replaced and got ready for the next casting. Repairs previous to the next charge are done in the furnace at the same time. Sometimes when the casting is made in a ladle the contents of the furnace are run into the ladle, which is placed directly over a larger slag ladle so that the entire contents of the furnace flows into the upper ladle, and any excess of slag flows over the side into a slag ladle below. Such practice is excellent for acid works also, as it recovers all the steel, and the disadvantage of having the sides of the ladle dirty is thus compensated for, by the reception of the slag in a ladle below, which puts it into a shape in which it can be easily handled. In the basic works there is an advantage in draining the furnace, but there should always be just enough slag left at the last to cover the steel in the ladle.

As soon as the casting is completed and the slag removed the bottom and sides of the furnace are repaired with the fine and well burned dolomite. Any excrescences are first removed by adding roll or hammer scale, and any depressions are then filled up with the "basic."* Gas is turned on to heat up the furnace and make the shrunk dolomite frit and unite with the rest of the "basic" on the hearth, and the tap-hole is closed. There is always a tendency for the bottom to rise. The additional material must be removed or fluxed off in order to prevent the diminution of the capacity of the furnace. It will take from one to three hours, depending upon the amount of repairs, to get the furnace ready for the next charge.

An example of the charge at the Alexandrowski works is given below :† In a charge of 6020 kilos. there were added 250 kilos. of lime, and 300 kilos. of pure iron ore. The average of phosphorus was 0.5 per cent. The time required to work a charge was 7½ hours. The result was 5390 kilos. of ingots. The first assay was made after the charge had been melted; it contained 0.458 per cent. of phosphorus. The second, taken after the melting of the scrap, contained 0.313; the third, taken after the addition of the ore, and shortly before tapping, contained 0.065; and the fourth, from a finished rail, contained 0.065. The slag had thus removed 0.393

* The fine shrunk dolomite is called "basic" by the men.

† Stahl und Eisen, vol. ii., p. 600.

per cent. of phosphorus, or 80 per cent. of the entire phosphorus contained in the charge.

The charge is generally made up of pig 30 per cent., steel scrap 20 per cent., large iron scrap 27 per cent., small iron scrap 23 per cent. One and one-tenth tons of this mixture is required to make one ton of ingots. The limestone added is generally about ten per cent. of the charge. The quantity of basic material used for repairs is about four per cent. of the weight of the charge, and the slag produced about ten per cent.

In order to make the repairs to the hearth it must be first very carefully cleaned of slag and metal, and then allowed to cool a very little. This is generally true, but especially so when magnesite is used. Neglect of this precaution often makes considerable difficulty. In some works but little attention is paid to it, even the draining of the furnace after casting being neglected, the repairs being made with some liquid steel, and the slag still remaining in the furnace. To properly make the repairs all the liquid and adherent material should be removed. The hearth is then carefully examined to find where the deepest depressions are. These are filled with shrunk dolomite alone, which gets hard in a few minutes. When this has been done the whole hearth is leveled up with the "basic." It is well beaten in, exactly as the sand is in making repairs to the acid hearth. When the whole of the bottom has been brought to good condition, the separation of the basic and acid materials must be carefully brought up to the proper state by additions made of the materials which are used, chrome iron, magnesia, magnesite, or bauxite. The greatest amount of repairs are usually required to the slag line, as the slag eats into the lining. This is done by throwing in the shrunk dolomite from the working door a little higher than the point required to be mended. The lining is sticky, and the material rolls down but a little way to the place to be repaired. The joint between the basic and acid material is made in the same way, but requires more care. The gas and air ports are repaired in Russia with coke dust and coal tar. They are sometimes worn when made of silica brick by the draft carrying lime dust against them which fluxes down the silica brick, but the best silica bricks made at Brymbo last as much as 150 to 200 charges. Lime bricks do not stand the expansions and contractions caused by the risings and lowerings of the temperature. Magnesia bricks made by hydraulic pressure might perhaps endure

these variations, but as yet nothing has been found that will bear them.

The tap-hole is repaired by running an iron into it and ramming dolomite thrown in from the working door against it. When completed the iron is withdrawn and the hole filled as usual with a mixture of 67 per cent. of quartz sand and 33 per cent. of fire clay, or with "basic," sand, anthracite, and clay, or it is stopped with the same mixture of sand and clay as in the ordinary process. The tap-hole requires repairs every three to four days; the hearth lasts 150 to 200 charges, and the roof and side walls and ports require repairs after every 30 to 40 charges. Generally, however, the ports require to be repaired twice as often as the walls. As the ports side of the furnace is the most thoroughly protected, it requires the least repairs. The regenerators require cleaning about once a month. These repairs will take from an hour to an hour and a half to make. The furnace has by this time cooled down somewhat. It will take about an hour to harden the new materials put into the hearth and to bring the furnace up to the temperature required for a new charge.

In Russia the construction of the furnace above the generator takes eight men seven days, with all the materials at hand, and costs about the same as an ordinary acid furnace. The construction of the movable roof of the furnace* would take less time and be much cheaper. A six-ton furnace requires 6000 Dinas brick and 1500 fire-brick; 6000 kilos. of ground dolomite, 1300 kilos. of dolomite in lumps, 1000 kilos. of chromic iron, and 1200 kilos of coal-tar.

The six furnaces in St. Petersburg,† in the year 1882, had 428 working days, made 834 charges, producing 6081 tons of ingots, using:

Cleveland pig,	1970
Old rails,	1409
Hematite pig,	303
Ferro-manganese,	35
Cast-steel scrap,	670
Other scrap,	2076
Spiegel,	577
	7040

* Journ. Iron and Steel Inst., 1884, part ii., p. 453.

† Stahl und Eisen, vol. ii., p. 603.

There were used:

Limestone,	453 tons.
Lime scale bricks,	120 "
Coal,	5503 "

The yield, in ingots, was 86.4 per cent.; in scrap, 2.6 per cent.; loss, 11 per cent. To produce 1 ton of ingots required 101 minutes, 905 kilos. of coal, and 1157.7 kilos. of material containing iron.

In the working of this process the phosphorus and the carbon go out almost together, the phosphorus commencing just as soon as the iron is pasty, while the carbon does not commence to go until it is melted. The manganese goes rapidly during the melting stage, then slowly for the first two hours, after which it remains about constant up to the last hour, when it increases a little. The silicon decreases a little during the melting, and after that undergoes almost no change. The sulphur remains constant from one end of the process to the other. This shows that both silicon and sulphur must be as low as possible in the metals used.

The process produces a high quality steel with, when properly worked, from 0.02 to 0.04 per cent. of phosphorus, from 0.01 to 0.03 per cent. of sulphur, and from none to 0.004 per cent. of silicon, with carbon varying from 0.06 to 0.45 per cent., from a material almost useless for any other process. No other process can deal so well with pig and scrap containing irregular quantities of phosphorus and silicon. The labor required is the same in amount, but more difficult to perform. The output is equal to that of the acid process, and the results just as certain, but the cost of the refractory materials and the repairs is greater. The difficulty of working, which has been very much exaggerated, has prevented the extension of the process, but it offers every advantage in making an exceptionally good steel from both pig and scrap, useless for any other purpose, as being too high in phosphorus for any other process, and too low in these same ingredients for the basic Bessemer.

I beg to acknowledge the special assistance I have received in preparing this paper, from Mr. J. Darby, of Brymbo, North Wales, and also to acknowledge my indebtedness to Mr. P. Gilchrist, of London.

THE INFLUENCE OF THE TECHNICAL SCIENCES UPON GENERAL CULTURE.

BY PROF. F. REULEAUX.*

No student of the world's present state of general culture can have failed to observe the potent influence which the technical sciences of our day are exerting, nor the extent to which they have fitted us for incomparably greater achievements than were possible a few centuries ago. Be it in the field of rapid transportation, at sea or on land; be it that we tunnel mountains, that we rise into the realms of air or delve into the bowels of earth; that quick as the lightning flash we speed our brain images to the antipodes, or carry the very sounds of our voices through the length and breadth of the land; or, viewed in another way, that on one hand we subject to mechanical service the mightiest forces, while on the other we let the innermost processes of nature, actions so subtile as to escape all ordinary observation, operate for and accomplish our own ends—everywhere in modern life, all around us, about us, with us, beside us, technical science is the busy handmaiden, the untiring companion, of whose presence we are only fully conscious when for a brief span her help is withdrawn.

But all this is well known, and even trite, and yet among the generality of educated people, and perhaps even within the narrower circle of professional men, it hardly seems to meet with the appreciation which it merits. The useful arts, scientifically developed, have not as yet been accorded a sufficiently extended and deserved recognition as a factor—aye, and more—as a powerful lever in elevating the standard of general culture. This lack of appreciation may be ascribed to the fusion of the technical sciences at certain levels with mere empirical knowledge of the arts, and conversely, to their frequently having sprung from such empiricism, and possibly also, to the fact that in their special sphere of activity these sciences abstain so largely from all ideality; that without the motive of gain, without the social ills which are still inseparable from industrial labor, they would not have found, would not attain today, their actual development. But interesting as this phase of the question is, it must not detain us now. Do not expect a panegyric on the technical sciences, nor a confutation of the argu-

* An Address, entitled "Cultur und Technik," delivered before the Industrial Assoc. of Lower Austria. Translated for the QUARTERLY, by Wheaton Kunhardt, E.M.

ments which would deny them their due acknowledgment. Both of these points are but of outward significance. Let us rather approach some of the weighty inner questions of the subject which appear to stand in need of special discussion.

In the first place: *What position do the technical sciences occupy in the active solution of the great problem of general culture?* This position has never been defined with half the precision that has marked our account of their social, political, and economic importance. A second question is: *What, in its leading features, is the general method pursued by the technical sciences in the accomplishment of their ends?* This method must underlie more or less distinctly all inventive effort, and the question is one which has excited, and will probably long continue to excite, lively interest among technists, jurists, and practical managers because of its connection with patent legislation. A third question that we may touch upon is: *What are the true ends and principles of technical education?* Investigation in this department has been very profitably forwarded by many professional bodies, but yet the subject has not ceased to furnish matter for controversy. If we consider the question here, it is because the answers to our foregoing queries may have an influential bearing upon its solution.

* * * * * * *

To compare the civilizations of other races with our own, we must evidently disregard all those communities that stand upon the lowest steps of the social scale, those, for example, that have not yet aspired to a written language; for among such the cultivation of the sciences is not conceivable. But over and beyond these limits we soon come upon great nations that for centuries, and even millenaries, have been the possessors of an advanced degree of culture. Such are the races of Eastern and Southern Asia—the Chinese, the Japanese, the Indians, Persians, Arabians. If we regard their civilizations without prejudice, we must admit that they are in many respects of a high order, and were so, indeed, when Central Europe still lay mired in the slough of barbarism. Even then sciences and the arts blossomed among the nations of the East, nor have they ceased in their development to this day. In most exalted form did the Hindoo worship the Deity three thousand years ago; a full two thousand years ago, and Indian poets had produced their nation's Odyssey, the Mahabharata, and dramas too, in rich abundance, and among these one, the Sakuntala, whose delicate charm remains ever fresh as its sources which well

from the depths of the soul. Philosophy flourished, and philology, too, even to such degree that the Indian grammarian of to day looks on an unbroken line of antecessors reaching upward to the idolized Panini. Mathematics, too, was fostered—do we not even now write our figures with Indian characters? And the industrial arts; how they flourished then, and how, in part, they flourish still in India, in Eastern Asia! And Persia; how brilliantly her poetry shone through ages! Her illustrious Firdusi, followed by her "Horace" of Shiras, Hafis, with his never aging songs, have both become endeared to us in excellent translations.* And, then, the literature of Arabia; what wealth of investigation has it not handed down to us; how profitable the cultivation of its Grecian heritage, and how it advanced astronomy, so that to this day we name half the heavens by its designations! And with what care did Arabians in Charlemagne's time, under tolerant and enlightened rulers, foster numbers and many far profounder sciences; and did they not, indeed, anticipate our own chemists in divers essences and principles?

Where, then, is the difference in intellectual sphere which has allowed a separation between them and us? Are we not confessedly their inferiors in sundry arts? Valor is theirs, nobility and justice are priceless virtues with them, even as with us. Where do we mark the points of distinction in a purely human sense?

Or let us rather put the question otherwise, if comparison upon intellectual domain cannot solve the problem which is certainly presented. Let us ask, whence is the source of our material preponderance over them? How, for example, has it become possible that England, with a few thousands of her own troops, should rule supreme over a quarter of a milliard of the natives of India? How was it possible for her in 1857 to suppress their terrible, fanatical revolt? How has it come to pass that we Europeans, or, not to necessitate a special mention of America, settled as it has been by Europeans, that we Atlantic nations are the only ones who have girt the globe with lines of railroad and of telegraph, and furrowed the seas with powerful steamships, and that to these great achievements the other five-sixths of mankind have contributed not a jot—the same five-sixths that are for the most part socially organized and in some cases highly cultured.

In various ways an explanation, or rather a definition, of this stupendous fact has been sought.

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* See "Characteristics of Persian Poetry," North American Review, April, 1885. —TRANSLATOR.

Klemm, the industrious Leipsic collector, an archæologist years before the discovery of the lake dwellings, proposed the distinction between *active* and *passive* races, and many at the present day still hold to his classification. In his conception we Atlanticists are the active races, and those others, downward to the entirely uncivilized, are the passive ones. We enact history, they endure it. This theoretic distinction, while seeming to have much in its favor, is none the less untenable. The chapters of history teach us that through long ages nations can be active, then passive, and then active again. Activity and passivity are not, therefore, inherent attributes of nations, but conditions into which and out of which they can come without essentially altering their intellectual status. But according to Klemm's view, with each of their transitions the nations altered their entire state as well. In fact, continual changes were going on, according as the adventitious fortunes of—let us say, secular—history superinduced them. This theory cannot stand the test of real experience. Europe might be subjugated, rendered passive, to-morrow by Asiatic hordes, without forfeiting those qualities which make the railroad, the steamship, and the electric telegraph her intellectual possession. The Arab might destroy the works of our technical science, as repute has it that Omar destroyed the books, without being able to reproduce, even in part, as he did, what his vandal hand had despoiled.* We must, therefore, discard Klemm's distinction, at least for all purposes of our research, as it fails to offer us any elucidation.

Some find in Christianity an efficient cause for the great difference that we have observed; but this, too, upon investigation, fails to meet the case. It is true that a considerable proportion of the discoveries and inventions which have shed their transmuting light upon our world of ideas was made in Christian countries, but by no means all. See in that brilliant galaxy the effulgence of the art of printing! Yet we know that the Chinese had invented it a thousand years before ourselves. The same is true of gunpowder, which was so potent an agent in the metamorphosis of our own civilization: Arabians used it long before the day of the Freiburg

* It is about time that we should cease from repeating the myth of Omar's destruction of the Alexandrian Library. In the first place, the capture of the city was not effected by Omar, but by his general, Amru. Further, the larger part of the library had long previously been destroyed—once, in the year 415, by a conflagration, which was fanned into flame by fanatical Cyrillians of the Academy at the time of unhappy Hypatia's murder; and earlier still, in the year 30 A.D., when the city fell into Roman hands, and when likewise a great number of the books were lost by fire.

monk. And in mechanics, the powerful waterwheel traces an Asiatic descent from far primeval times. These are merely illustrations. But turn to a true offspring of Europe's brain, the steam-engine, and note in its gradual development into practical shape the activity of the renaissance in Italy, Germany, France, and England, yet nowhere else in Christendom—not, therefore, Christianity identified with that progress, but on the contrary, her priests too frequently throwing the whole weight of their authority against its onward course. But let us look still further. Are there not Christians in the East to-day, in Armenia, for example, and throughout the broad extent of Abyssinia, who are living entirely without the pale of our victorious modern science? Naught have they contributed towards it, nor aught will. It is not the things, therefore, not the inventions in themselves, but the ideas, the thoughts which accompany them, that must have called into being the great cultural transformation.

We cannot, indeed, do otherwise than attribute the change to a remarkable progress in the intellectual process; a difficult, hazardous ascent to higher and freer interpretations of nature. The idea was conceived, soon to burst all fetters that would bind it, that nature's forces in each of their manifold effects obey not the mandates of an ever intervening—a divine—Will, but act by the governance of immutable laws, and never otherwise.

> "*Nach ewigen, ehernen,*
> *Grossen Gesetzen*
> *Müssen wir alle,*
> *Unseres Daseins,*
> *Kreise vollenden,*"*

so thrills the inspiring genius of Gœthe in presence of nature's inexorable powers. But also, *nach ewigen, ehernen, grossen Gesetzen* great worlds revolve, and stars in their distant orbits course—falls the roof-tile, too, from the weathered eave, and the rain-drop from the cloud.

> "*Sonnen wallen auf und nieder,*
> *Welten geh'n und kommen wieder,*
> *Und keim Wunsch kann's wenden!*" †

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* "According to great Adamantine laws, Eternal, unchanging, Must we all Our earthly being's Cycles complete."

† "Suns are fading, suns are glowing,
Worlds are coming, worlds are going,
And no will can stay the change!"

Thus in admirable poetic garb has the soul-fraught prelate, Hebel, informed the consciousness that not a material, but a spiritual, world compasses the apprehension of God, and that in the very immutability of its laws lies the grandeur of the material creation. To lay hold upon this idea thought broke down the old barriers, nor did it then tarry long before making its deductions for the practical exigencies of life. The conclusion that was reached, when stated in the abstract, and freed from all secondary matter, was this: *If inanimate bodies be brought into such situations or conditions that their action under natural laws will subserve our purposes, then we can let them perform work for, and in the place of, living beings.*

Conscious practice on this basis has created modern technics. Technical science I have called it, and must abide by the name. For those very laws of nature which intelligent endeavor was striving to apply were scarce known when the intellect of Europe entered upon its new path. They had, for the most part, still to be unfolded, and to obtain knowledge became the burden of an arduous strife. For the world of letters believed at the time that it possessed the hidden secret of these laws, and so it was that the innovators had not merely to make the discoveries, but to dethrone old, opposing convictions—verily a herculean task, but at once a militant movement toward the heights of intellectual freedom. This advance encountered vehement opposition in the prejudices of the Church. Many a milestone of its progress is marked by bloody sacrifice. Galileo's bent form here rises before us: what if he did not pronounce the memorable *e pur si muove*, as the micro-historic investigation of recent years pretends; the thinking mind of Europe spake it for him. The victory was gained and the domain for our modern technical science conquered. To-day the old-time reaction has spent its passion; small skirmishes may still occur in the rear guard, the results of obstinacy rather than conviction, but at all events the great onward movement is not retarded in the least.

What would have happened had reaction vanquished—for reaction indeed it was, since Germany had begun the advance over a hundred years before, and Copernicus had lain in his grave for more than ninety years when Galileo was forced to testify against him—what would have happened, it is hard to conceive, or rather no, we can see it, we can note the very thing in the fate of the great Arabian family of nations. There reaction had really carried the day. The Galileos of the East, the Averrhoës and un-

numbered others, men of free conviction all, were completely overthrown, and with their fall Arabian culture, just stretching forth her hand to grasp the trophy of freedom, was stricken down by the victorious fanatic, and there she has lain, stunned, paralyzed in every limb, for half a thousand years. *Allah aalam!* "God only knows!" And hence—"nothing is that thou must want to know!" Such has been the pure Mohammedan's creed ever since; all investigation is denied him, is declared a sin. A refined and noble moslem* expressed the hope not long since that even yet his people might be called to the front to take up the lost banner of leadership. But who can possibly believe him? † It seems but too certain that the defeat of intellectual enfranchisement in all lands of the Arabian tongue has sealed the doom of the other Asiatic civilizations. Like a broad dam the heavy mass of deadened mind lies between them and us, and so it has come to pass that we alone have trod that path of progress which was pioneered by the great intellectual evolution which I have described.

The forces of nature which that advance taught us to look to for service are mechanical, physical and chemical; but the prerequisite to their utilization was a full equipment of mathematic and natural sciences. This entire apparatus we now apply, so to say, as a privilege.

For the convenient designation of the two systems which we have recognized it will serve us to select a special name for each. A penetration into nature's secrets was revealed at quite an early day among the Medes and Persians, and especially among the tribe of the Magi, who earned thereby so great repute that their name devolved upon a priesthood caste. Even the Greeks were so much imbued with an appreciation of their knowledge, not to mention the fear it created among the ignorant, that they called any skilful device, or any arrangement by which extraordinary results were achieved, a magic work, or, accommodating the word to their tongue, a *manganon*. All kinds of contrivances cleverly

* The Afghan sheik, Dschemnal Edin, according to whose own testimony in the *Journal des Débats* of 1883, "the caliph, Al Hadi, put to death 5000 philosophers in Bagdad in order to exterminate science, root and branch, in all Mohammedan countries.

† One can hardly do otherwise than regard the case as utterly hopeless, for the Arabians have lost the national unity which alone could infuse them with a sufficiently strong initiative, and because their whole family of languages has become so dismembered that all thought of reunion is now indeed futile.

conceived and deftly wrought were so styled, and among others, a catapult, a powerful engine of war, with which the word travelled down into the Middle Ages.* Then, early in the seventeenth century, when large machines were invented for rolling and smoothing linen, and the apparatus happened to possess marked outward resemblance to the ancient catapult, that accidental circumstance gave it its name, and this soon passed into all the languages of Europe, as every housewife knows—or possibly does not know—when she sends the domestic linen to the "mangle." † Let us rehabilitate the old word for our purposes, and designate by the term *manganism* the utilization and control of the forces of nature through their own recognized laws, while the opposite aim, which remains set against the study of nature, or at most, in ways mysterious and beguilded, catches a few whispers of her secrets, we may call *naturism*.

Adopting these names, we see, then, that all nations, or in fact, all orders of civilization, can be classed either as manganistic or naturistic, and we have already remarked how the former, by developing their material resources upon a broad basis of knowledge have far outstripped the latter in the march of progress. Indeed, we need not hesitate in asserting that to the manganistic nations belongs the empire of the earth. Now, as in all times past, war is waged for this dominion, but every observer can predict with certainty that the manganist will issue victorious from the conflict, or that those nations which cannot resolve upon adopting manganistic tenets must face the alternative of subjection and decay.

That with set determination it is not impossible to pass from naturism to manganism, we can see to-day in the action of Japan. With keen insight the Japanese appreciated the truth of their position, and they are now attempting the feat, without parallel in history, of passing almost at a stride from naturism to manganism. The clearest-headed thinkers of the nation have understood the

* The *mangano* of the Italians, and *mangan* of the French.

† Zonca, *Novo Teatro di Machine*, Padoua, 1621, p. 34: "Mangano in questo luogo si dimanda quelle Machine con laquale si lisciano et lustrano le tele, ciambolette, rasse et altre cose, che nell' arte militare, il Mangano s'intende uno stromento per lanciar armi, pietre et altre materie, come si puo vedere presso d'alcuni Auttori." It appears, then, that in 1621 the catapult was already antiquated, though still holding a place in the books. The illustration shows a mangling press more than thirty feet long, and driven by two men in a tread-wheel; another machine is described as being driven by horse power.

necessities of the case and have been able to win over the political powers for their cause, and so we behold a clever and sensible people altering their entire system of education, and summoning their full strength for an earnest engagement in the new undertaking. Difficult as the task may be, yet its beginnings presage success, for these have consisted in a constant learning, learning, learning!

Manganism cannot be bought; it must be a product of education, a growth. The demonstration of this is fulfilling itself in China, where all the purchases of most improved European munitions of war seem to avail nothing in face of the systematic attacks of the manganists. The naturists of the Celestial Empire probably overlook the fact that mere drilling in will not suffice; a single minute detail in a process, omitted or forgotten, may endanger the success of the whole. Many may perhaps remember the little mishap which befel the Japanese ten or twelve years ago when, jealous of their independence, they dismissed their English instructors from the first of their large men-of-war, and forthwith undertook great evolutions with her; how beautifully, too, they manœuvred, but how, finally, to the amazement of all spectators, they continued steaming for hours in wide circles around the bay. The engineer had forgotten his last potent "sesame;" he could not recall how to get rid of the superfluous steam in his boilers, and so the vessel was kept under way until the entire steam supply had been exhausted. To-day the Japanese know better, and laugh as heartily as any over that first trial.

In India the English have, in a quiet and unobtrusive way, initiated a movement toward manganistic methods of education. All endeavors, however, are as yet in the bud, though some day they may ripen into fruitful bearing.*

* In a clever essay entitled "*Islam and Science*," Ernest Renan shows by an interesting illustration how deep and strong a root naturism has taken among the nations of Islam, the heirs of a once glorious Arabian culture. The English Nineveh explorer, Layard, had addressed himself to the Kadi of Mossul for certain data concerning the population, commerce and history of the city. The Kadi replied by letter in this wise:

"Oh, my illustrious friend, oh, joy of the living! What thou askest of me is at once useless and hurtful. Though I have lived all my days in this land, yet has it never entered my mind to count its houses nor to trouble about the number of the dwellers therein. And to your question how many goods this one may load upon his ass, or that one into his boat, such matter indeed is no concern of mine. As for the history of this city, God alone knows it; He alone could say with how many errors its in-

But we need not go to distant lands in quest of naturism, for here in Europe we may find it still; indeed, we can detect a trace of it in every living man. It is only through education that manganistic modes of thought ally themselves to man's own self; that the higher rational faculties and dispassionate logic are linked to the naïve and to whatsoever in man is fair nature's unconditional devotee; but then this intellectual balance gives growth to calm deliberation, and enduring steadfastness against assaults of nature that threaten ruin—in a word, it develops a type of character absolutely antagonistic to fatalism.

In Spain the development of manganism has been insignificant. The Iberian peninsula did not contribute to the great, revolutionizing inventions, and we must assume that any expression of reform sentiment there could be the more easily suppressed because all popular interest was directed toward the newly discovered western world. What Spain has lost by her standstill is beyond all power of calculation.

Greece, once at the bright summit of attainment in sciences as in arts, found herself, when the day of modern technics dawned,

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habitants were filled prior to the day of Islam's conquest. For us the wish to name them would be dangerous.

"Oh, my friend, my lamb! Seek not to know what concerns thee not. Thou camest, and we bade thee welcome; depart in peace! In very truth all the words thou hast spoken have not pained me in the least; for the speaker is one, but the listener is another. After the manner of the men of thy people, thou hast wandered through many lands, yet nowhere hast thou found happiness. But we, God be praised, we were born here, nor have we any wish to depart.

"Listen, my son, there is no wisdom like unto a belief in God. He created the world. Is it for us to strive to equal Him by searching into the secrets of his handiwork? See yonder star which circles in the heavens about yonder other star; mark again, still another star, drawing in its wake a tail, requiring so many years to come and then so many years to go. Leave it, my son; He whose hand fashioned it will surely guide it.

"But perhaps thou wilt say: 'Oh, man, withdraw, for I am more learned than thou; many things have I seen of which thou knowest nothing.' If thou believest that these things have made thee better than I, then I bid thee doubly welcome; but I thank God that I search not where I need not know. Thou art versed in things that are indifferent to me, and what thou hast seen, I despise. Will thy farther reaching knowledge procure thee a second stomach, and thine eyes which have cast their glances everywhere and have searched through all, will they espy thee a paradise?

"Oh, my friend, if thou wouldst secure happiness, cry aloud 'God alone is God.' Shun evil—then wilt thou fear neither man nor death, for thine end will surely come."

still so crippled by the fall from her high estate, that the new born spirit could not infuse her. But to-day her people are striving to elevate themselves out of their naturism, and with interest we may follow the efforts made upon the classic soil of that fair land to recover the old heritage of intellectual vigor. Yet without a method, based on manganism, all endeavor will be in vain.

Italy offers us a remarkable spectacle. Her masses, deeply devoted through ages to the spirit of naturism, even after her active participation in the great scientific discoveries of the renaissance, have eschewed well nigh completely the younger genius of manganism, but the flower of Italian art has ever been kept in bloom, and therein her gifted people sought and found their country's fame. But now, in the day of her political regeneration, Italy feels the necessity of repairing the neglect of centuries, and so we see the whole country applying itself with astonishing energy to spreading wide the introduction of manganistic industries and a knowledge of manganistic capabilities. It cannot be disguised, however—and the recent national exposition at Turin seemed to emphasize the fact—that the rapid and really significant industrial progress of the country has tended to weaken its productiveness in art.* In making such observations it seems indeed—and like a cloud the thought shadows our reflections—as though a trenchant antagonism separated the two aims, demanding the sacrifice of one. But happily such is not the case. Fine arts and scientific technics do not preclude each other. But justice to both calls for all the stronger effort—greater firmness, deeper intellectual absorption in the refined laws of æsthetics, to repel the shocks of the machine's disturbing onslaughts. That both may flourish side by side bear witness the active movements of each in Austria and Germany to-day.

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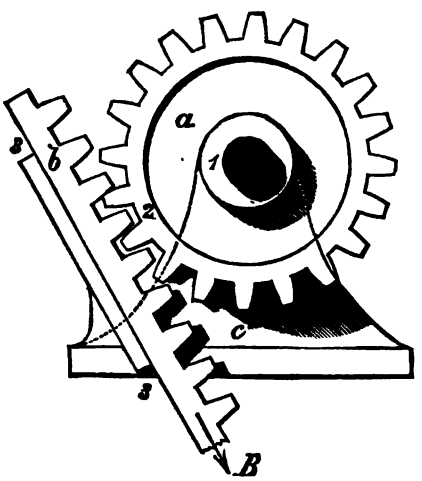
In turning now to a consideration of the method of manganism, a whole series of familiar introductory steps may be passed by, and, for the sake of brevity, we may confine ourselves to dwelling upon certain fundamental principles common to widely different and

* In conversation with an eminent Italian, a man of acute perception, the latter said on this very point: "Speak your thought openly, and say we have fallen back in the fine arts. We are well aware of that ourselves, nor do we grieve about it either. First we must retrieve in the other direction what we have lost through long neglect, and for this work we require our best brains. Later on we will manage to regain our old standing in the line of art."

quite distinct operations. The crowding of new phenomena upon the technical field is so tremendous, and the difficulties thereby cast in the way of including them in any of the accustomed categories so formidable, that only a comprehensive theoretic generalization can command a complete survey. The formulation of such an expression presents no difficulty, but in order to obtain full bearings on the subject, it will be well for us to consider the following simple cases from which important definitions issue:

In Fig. 1, let us assume that the pinion *a*, which engages at *2*

FIG. 1.



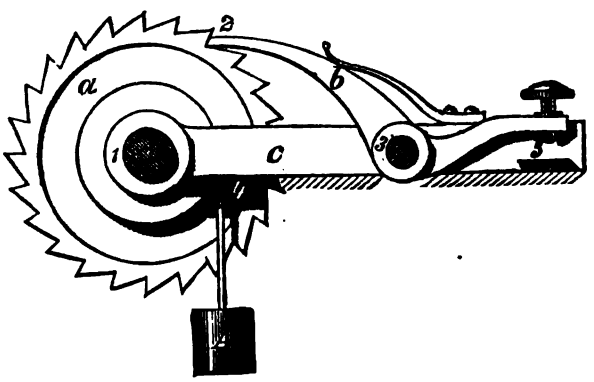
in the rack *b*, can rotate upon its axis *1* in the stationary support *c*; assume further that the rack, which we will suppose to be of considerable length, slides in the same support at *3*, and that a weight *B* is suspended from it. If the wheel *a* be so revolved that *B* is raised, or lowered, the combination will be an example of a train of mechanism, that is, of a piece of machinery capable of a continuous transmission of motion, either forward or backward. Let us term this, because of its continuous transmissive action a *Laufwerk* *—or transmitting element. There are any number of

* This is the first of a series of terms which the author has employed, and in part coined, with consummate skill, for which no equally comprehensive and at once concise expressions exist in English. The *werk* of the German compounds is rendered in literal translation by "work," in the sense of its occurrence in such terms as wheel-work and link-work; it conveys the idea of a combination, usually mechanical, but extended in this essay to embrace certain physical and chemical combinations as well. Considered in the author's meaning as a compound unit, the word "element" has been selected as possibly the most suitable, though not a literal, nor an exact, rendering for it.—TRANSLATOR.

familiar transmitting elements in which shafts, friction and belt pulleys, gearwheels, cranks, etc., are employed in various assemblages.

Distinct from these is another class of mechanisms characterized by quite a different kind of motion. Fig. 2 will furnish us with examples:

FIG. 2.



The wheel *a*, Fig. 2, is again assumed to rotate in a stationary bearing about the axis *1*; it is armed with teeth in which the ratchet *b* engages at the point *2*. These teeth prevent the wheel from yielding to the impulse of the weight *A* which is suspended from it. But if the wheel be so revolved as to wind up the suspending cord, the ratchet will not interfere with this action, though it remains ready at all times to prevent the wheel from reversing its motion upon a cessation of the impelling force. The arrangement may be designated as a *Sperrwerk*—or engaging element. Its motion is intermittent, and moreover, the positive and negative movements are of different character, so that the mechanism is quite distinct from the transmitting element.

But the engaging element is by no means the only one that can be derived from our combination; no less than five others are possible.

Suppose, to begin with, that the ratchet *b* be released by pressing upon the button at *5*; the weight *A* then drops, and in its fall rotates, or drives, the wheel *a*. The motion thus impressed upon the wheel can be variously utilized—either all at once, for percussion, as in a ram, or gradually, as in clocks, or intermittently, according to need. In every case the mechanical work that has been stored by raising

weight, an elastic body, a spring, for example, can be brought into the weight can be usefully employed. But instead of raising a tension. We may therefore call the arrangement a *Spannwerk*—that is, a tension, or storing, element.* The old cross-bow was such an element, and millions of them are used to-day in gun-locks.

We obtain a third mechanism by a slight modification of the procedure, namely, by allowing the ratchet to reëngage in the wheel after its first release. It then stops the wheel and so arrests the weight in its fall. The mechanism can therefore serve for checking falling bodies, and hence its name, a *Fangwerk*—or checking element. The safety catches employed for arresting the fall of cages and elevators in cases of rupture of the hoisting cables are such elements. But the teeth of the wheel *a* may be reduced to extremely small dimensions, and even to the point of vanishing altogether; the serrated periphery then passes into the form of a smooth rim, the ratchet becomes a body in mere sliding contact, and the whole mechanism is transformed into a frictional catch. Thus it appears that brakes of all kinds, the common train-brake included, are species of checking elements, and that applications of this kind of mechanism occur frequently and are extremely useful.

A fourth mechanism is developed from our combination by the addition of a second ratchet, similar in general form to the first one, but attached to the end of a swinging arm. By the vibrations of this member, the wheel can be intermittently revolved so as to effect a gradual raising of the weight; with each forward stroke it is turned through a certain arc, while during each return stroke the fixed ratchet prevents it from reversing its motion. The mechanism formed and driven in this way is known as a *Schaltwerk*—an intermitting element. Its applications in practice are not uncommon; the automatic feed motion of many machine tools is a familiar example of its use.

A fifth modification of the arrangement arises in the employment of merely a small sector of the wheel to block the passage between the points *1* and *2*. By closing the gate, or lock, at *2*, all passage is prevented, and only a subsequent release can reëstablish it. In this particular application the mechanism is a *Schliesswerk*—or blocking element. It is used in millions of locks for

* Or, more nearly, a potential element.

windows, doors, chests, etc., and frequently in connection with the intermitting element.

The sixth, and from a mechanician's standpoint perhaps the most remarkable mode of applying the combination, is in form of an escapement, or *Hemmwerk*—a releasing element, as we will call it. It is produced by tapping lightly on the button at *5*, releasing the engagement for a moment and then, almost instantly, closing it. When this movement is carried out with perfect regularity, the progression of the wheel *a* can serve for measuring intervals of time. Releasing movements fulfil this important function in untold millions of clocks, the uniform release always being effected by means of an organ of isochronous vibration, such as the pendulum or the balance-wheel. But besides this application of the element, many others occur in practice.

Thus we see in these six movements, these six *Gesperrwerken*—or interfering elements, as we may collectively term them—the solution of a vast number of practical problems. Yet our survey of their applications is far from complete. We find upon continuing our examination, that these elements are frequently combined with one another, the action of the first being transmitted to the second, and so on. A pretty illustration is afforded by the hair trigger of a target rifle. This device is no more than a small storing element which is very easily disengaged, and the potential energy so liberated acts upon and releases the firmer storing element of the hammer.* This is a case in which one storing element disengages another. We may call such a combination a storing element of a higher order, or, in general, when any interfering element is combined with another in such a way as to impress its action upon it, we may speak of an interfering element of a higher order. Such, by way of illustration, we find in the works of a clock, in which either a gravity, or spring, storing element actuates the releasing element of the clock, operating therefore in second order. Manifestly we are here confronted with a principle—for the action can be impressed equally well by an element of interference upon one of transmission, and by that one upon any other. In fact, in the example of the clock we have just had before us, the releasing element actuates a spurwheel gear—in other words, a transmitting element—which in turn moves the pointers. If we agree upon

* It may be remarked in passing that a hair trigger of highly developed form was used in cross-bows toward the end of the Middle Ages.

the term *Treibwerk*—or driving element—as generic for any simple or complex movement gear * whatever, then the works of a clock constitute a driving element of the third order, compounded in ascending order of three simple driving elements of storage, release and transmission.

After this expansion of our horizon, there is yet a further development for us to seek, an extension in another direction. If we consider the many machines in practical use, we observe that fluids serve for the translation of motion in quite a number of them—in the hydrostatic press, for example, as also in the pump, and hydraulic engine, and in waterwheels, turbines, etc. Not liquids only, but gases are similarly employed, in gas motors and air engines, and, especially, in the steam engine. Closer investigation reveals that in all these cases the fluids are so constrained to move by means of inclosing ducts and vessels that they act in suitable machines the part of solid bodies, but always with the advantage of conforming perfectly to the varying shape of the confining sides.

Let us apply this idea to the transmitting gear illustrated in Fig. 1. By substituting a stream of water for the rack the element becomes a waterwheel, if the water acts as driver, or a lift, or flashwheel, when the wheel is the driving body and water the driven one.

Machine practice applies the same process of substitution to our several interfering elements. If a fluid fulfils the functions of the wheel *a*, or of any of its equivalent sectors, racks, or other modifications, then the ratchets are called valves. Valves in fact, however they may be regarded, are nothing but the ratchets of fluids. What a new, what a grand, extension the application of the driving element now acquires! Illustrations abound; they fairly crowd upon us. Our ordinary pump with its valve-piston and suction valve is a hydraulic intermitting element, developed on precisely the same lines that we followed in forming such an element from Fig. 2. And as for hydraulic storing elements, they are coming more and more into use every day, particularly in the form of so-called accumulators.

In releasing elements, too, fluids have been put in the place of the balance-wheel, or of its equivalents, as we see in all hydraulic engines and in the steam engine. Considered purely as driving elements, these machines correspond with the clocks which served

* *Bewegungswerk.*

anon as our example of a releasing element, the only substantial difference between them being that the releasing element of the clock overcomes none but hurtful resistances, whereas engines overcome mainly useful resistances. If our limits permitted, it would be an easy matter to mark the points of correspondence throughout. Take the valves, for example, which frequently occur singly, but also combined in twos and greater numbers in a single piece —the slide-valve of the steam engine is the anchor escapement of the clock; the pendulum of the latter is the reciprocating piston of the former, and so on throughout. See how the mighty steam engine ranges itself into line and modestly occupies its proper station. Thus should it always be in a scientific conception, for therein neither sensation nor aught else save true logical connection avails.

But still we have not quite finished with this machine nor with this principle. One final ascent has yet to be made ere we can command a perfect, theoretic horizon. If we consider the source of power of our steam engine from a general standpoint, we find that the living force imparted to its molecules, which we call heat, is intensely active, and that by virtue of this activity there resides in the aggregate mass of stored up steam a certain tension. The steam boiler, therefore, with all its valves and fittings—what is it other than a tension, or storing, element, and one that differs only from those of our former consideration in that here the tension is produced by physical means. In the steam generator, then, we recognize a physical storing element. This idea soon leads us further; it draws us onward to seek, in the line of causal connection, whence the heat in the boiler water. This we find in the furnace fire, in the glowing, flaming fuel which disengages by combustion, through chemical action, the thermal energy that it has stored. The fire, then, is the dissolution, or running down, of a chemical storing element which, when released by the act of ignition, disengages with sudden vehemence the thermal energy that, in the case of coal, was stored within it æons ago by the slow alchemy of nature.

Now we know our steam engine in its completeness, for we can clearly recognize:

In the furnace fire, ...	a released chemical storing element;
In the steam boiler, ...	a physical storing element brought into tension by it;
In the engine proper, consisting of cylinder, piston, and valve gear,	a mechanical releasing element driven by the preceding one;

taken as a whole, therefore, and neglecting all merely adjunctive parts, it is a driving element of the third order. If in the place of a simple engine with reciprocating motion only, we have an ordinary fly-wheel engine, then the releasing element is supplemented by one of transmission in the form of a crank gear which we utilize in thousands of familiar ways. In this, its commonest type, the steam engine therefore proves to be a general driving element of the fourth order.

One other illustration from steam engineering is particularly instructive: let us glance for a moment at the railway train. In the locomotive engine we have, according to our last development, a driving element of the fourth order. To this we must add the driving wheels as a frictional transmitting element, and to this, in turn, as a second transmitting element, the moving train upon the track;* the train of cars and engine together thus constitute a driving element of the sixth order. But suppose our train to be a modern one, equipped with the Westinghouse brake. Why is it, indeed, that this appliance has become so popular and so important? Our theory explains it at once. The brake itself is a checking element which had formerly to be set in action by hand, but note how different our procedure is to-day. We have at our command in the Westinghouse apparatus a powerful and ever ready storing element with which, in form of an air receiver, each car of our train is provided. By the movement of a single valve, in which we recognize the transformed ratchet of our earlier descriptions, this multiple storing element is easily disengaged and allowed to act upon an air piston, that is upon a releasing element, which in turn acts upon the brakes. Examining now the complete assemblage, we recognize—

In the small steam engine, a . .	. *Hemmwerk*—or releasing element,
" " air compressor, a . .	. *Schaltwerk*—or intermitting "
" " air reservoir, a . .	. *Spannwerk*—or storing "
" " air piston, a . . .	. *Hemmwerk*—or releasing "
" " friction brake itself, a .	. *Fangwerk* —or checking "

and in the whole appliance a mechanical *Treibwerk*—or driving element—of the fifth order. If we extend this combination, as we properly should, so as to embrace the steam boiler and

* In a cog-wheel railroad, the guide rails are not identical, as in this case, with the rails used for the transmission of power, but a separate power rail and power wheels are provided between the regular guiding tracks.

furnace, the apparatus develops into a general driving element of the seventh order. Assemblages of still higher orders are by no means of uncommon occurrence.

We are now in a position to explore without fear of confusion the newest of technical domains, the field of electro-mechanics. Here we find in the galvanic battery a chemical transmitting element—for we may permit ourselves the use of the term, seeing that a transmission of motion, albeit molecular motion, is involved. The physical current of electricity, whose valves or ratchets now take the form of contacts and brushes, is used in manifold ways. In telegraphy it acts as an intermitting element of the second order in the relay battery, starting and stopping a mechanical transmitting element and operating the recording pencil, so giving rise to combinations of the third and fourth orders. The ordinary railway bell signal operates in the fifth order—chemically, in the generation of a current; physically, as an intermitting element, by the attraction of an armature, which effects the liberation of a storing element that has previously been brought into tension by hand; then mechanically, in the operation by this latter of a releasing element which in its turn sets and releases the small spring hammers that form the final storing elements.

Among the chemical *Treibwerke*, or driving elements, it appears that those whose function it is to store energy occupy a very prominent position. Those among them that are artificial are so prepared by the chemist as to liberate their energy slowly or quickly, according to his purpose. Gunpowder is the powerful chemical storing element with which the groping naturism of mediæval Europe replaced the lesser mechanical element of cord and bow, that depended for its energy upon the unaided action of human muscle. The purpose remained the same; only the species of the storing element was changed. The tinder which liberated the new element was itself a slowly disengaging chemical storage element, entirely distinct from the greater one. At a later period inventive genius incorporated both in a single contrivance, first in the flint-lock, and then in the percussion-lock, entering in this way upon combinations of third order. The percussion cap, as an easily liberated chemical storing element, could be readily released by the mechanical element provided in the gun hammer. The ball was projected, therefore, by a storing element of the third

order, or, in the case of a hair trigger lock, with one, even of the fourth order.

To conclude our survey let us devote a few words to that small but very interesting object, the common match. How short, indeed, is the period, scarce as yet two generations, during which we have possessed it! And prior to that time we manganists stood, in point of lighting a fire, on very nearly the same level with the humblest naturist. The latter, as we know, starts his fire by a dexterous and laboriously acquired manipulation of two pieces of wood. In other words he releases the very stable, or difficultly liberated storing element, fuel, by the immediate act of friction. The method of the ancient Greeks was the same.* Flint and steel were the invention of a later day—a physical storing element which was applied to igniting a specially prepared and easily liberated chemical storing element, tinder, and this, as soon as it glowed, was used to release a more stable element, the sulphur string, with which, finally, thin slivers of wood could be lighted. Four superimposed storing elements were involved in the process: one of them physical (the flint and steel) and three chemical (the tinder, sulphur, and wood).

The philosophy of the match lies clearly within the limits of this evolved principle. That important little instrument was the result of combining at first three, and subsequently four, storing elements. It is a chemical storing element of the fourth order, comprising the elements phosphorus, potassium chlorate, sulphur, and wood. More recently paraffin and wax have been substituted for the sulphur, but the same principle is maintained and may be clearly recognized. Each of those storing elements that follow upon one another is more stable than the one it precedes, but its release in due turn is accomplished with certainty; and hence, as ultimate effect, a light and even delicate, mechanical action upon the first, most sensitive element, the hair trigger, so to say, of the whole,

* Their instrument was the pyreíon. It consisted of two parts: a lower piece, or eschára, which was bored with a hole, and a trypanon, or borer, which set into it and was rapidly twirled around. Could not the pyreíon, perchance, have been preserved in some hidden nook of the Grecian hills? Worthy, indeed, would be the service that should bring it to the light of day. The small tinderbox, which many of us can recall from earliest childhood, with its contents of flint and steel and sulphur thread, must surely have escaped utter extermination by the little match, and it were well to preserve it from absolute extinction by donating the few remaining specimens to ethnographic museums.

effectuates the release of that last, very stable engagement which once caused so much trouble, and made such arduous demands on the strength of one, or even two, active men. That the combination of the four storing elements was not realized until so late a day argues that the underlying mental process must have been a difficult one.

At last we have obtained a complete view of the manganistic principle, not in general outline only, but in the detail of great examples illustrating the activity of the most powerful forces, as also in cases involving the minutest reactions, and now we are able to state that *the method of manganism consists in the development of mechanical, physical, and chemical driving elements upon the basis of a scientific knowledge of natural laws, and in their subsequent combination in super- and juxtaposition.*

Although this principle has been unfolded with reference mainly to mechanic arts, it permits of an equally ready application to the processes of technical chemistry, and so we may regard it as embracing the problem in its entirety.* A single thought directed to the manufacture of sulphuric acid, or the coal tar colors, or any other chemical products, suffices to remind us of the mechanical and physical elements which operate there as intermediates, just as in mechanical combinations the other two classes were frequently found to enter.

* * * * * * *

From the height of our newly gained vantage-ground, let us look once more upon the technical sciences, and in their every feature and detail we can perceive how closely their results are interwoven with all our habits and forms of life, indeed with our entire civilization. We need not here take cognizance of the fact that in our dwellings we are fairly surrounded by thousands of interfering elements† which have made our space enclosures what they are—secure and comfortable habitations, furnished with warmth, and light, and abundant air. All these features we may disregard, because naturistic practice might achieve similar, though less perfect,

* Modern chemistry has applied itself with remarkable success to the production of chemical storing elements of very high energy and instantaneous and complete release, that is, to the preparation of explosives. But the contrasting problem is also conceivable; the formation of storing elements of high energy but gradual release. As heat diffusers, with a mild and vivifying, instead of a destroying influence, they might be of vast importance.

† *Gesperrwerken.*

results. But look to other features which impart a character to our abodes. There is the gaslight in each house, and on the street, and in the public buildings. We owe it to a chemical storing element* of the fourth order (fire, retort, gasometer, conduit and cocks, neglecting all the secondary parts), and all of this wonderfully and systematically ramified through the town. The water supply for household and public use, when derived from a river, furnishes us with a driving element† of at least the sixth order. On our railways we are conveyed by driving elements of high order, whose service we regulate by means of others equally high; our freight is moved from place to place, from land to land, from continent to continent, in quantities greater a thousandfold than man and animals could carry. With a physical driving element we conduct a written and spoken message service the wide world over. And how do we proceed in war? In millions of chemical storing elements, great and small, and almost all of them of an advanced order, we carry potential energy out upon the distant battle-field, and there release it by means of driving elements of high combination. Out upon the broad ocean, hundreds of miles from land, the energy of a storing element, which we derive from a gift of nature, operates through driving elements of high order to carry us for weeks and months through wind and wave.

Highly potentialized storing elements, like coal, have been eagerly sought out in nature—at an early day naturistic man found the water-course, the subordinate element of transmission for the greater storing one in the mountain heights—and in future we may still find others, as petroleum was found, rediscovered we may say, three decades ago. In this nature provided us with a highly energized chemical storing element, particularly adapted to disengagement when its constituent particles were at a bright red heat. It really consisted of a number of storing elements, in some of which the engagement was so slight that their release was apt to occur unintentionally. We were obliged, therefore, to subject the natural product to a process of separation, which, by the manganistic means of fractional distillation, removed the groups of particles that were too unstable, and then only did we obtain a storing element that could be safely transported and come into general use. The restrictive police ordinances required, as we see, that the engagement should be a firm one, and how favorable

* *Spannwerk.* † *Treibwerk.*

in their action these regulations have proved, the results to-day abundantly testify. This liquid storing element which was obtained, so to say, almost ready made from nature, replaced for illuminating purposes a number of others which we had derived from the organic world with the aid of considerable manganistic machinery.

Let us, before concluding, direct our attention to another point. In a conflagration we see an unintentional release of chemical storing elements—the class to which so many bodies prove to belong. The ratchet has been raised against our will; with increasing velocity, frequently at raging speed, the tremendous storing element runs down. But we hasten to check its course with another driving element, once worked by man-power alone, but now just as commonly operated by a chemical storing element (fire and boiler) through the intervention of a mechanical driving element of high order. Sometimes, too, we apply the action of a chemical storing element directly upon the quenching water, as when we use a gas fire-engine, or chemical engine, as the Americans are wont to call it. In this latter case, the driving element for the water is of a much lower order than before. It furnishes an illustration of how driving elements, whose ultimate effects are the same, compete with one another by rival endeavors to reduce the magnitude of their ordinal coefficients, or, in other words, to diminish the number of simple elements in their combinations. Everywhere, therefore, the manganistic thought, the manganistic principle, with which we sustain so largely, and ease, and defend our lives, but with which, too, we accomplish so often the destruction of our fellows.

And, finally, our industries, the producers of all our articles of daily use, and in turn again, of the manganistic machinery we require—what powerful influences have they not exerted through manganism in promoting our general culture! Just here let us pause, to enter the subject a little more in detail, and fix, if possible, upon a unit or standard of measure.

An essential factor for manganistic work is coal. It is now mined to the extent of about 400 million tons per annum, and is used mainly for industrial purposes. The amount produced in excess of this quantity suffices to cover the consumption for general heating. We have, therefore, for each of the 300 working days in the year, 1⅓ million tons of coal which are applied to chemical, mechanical, and physico-technical purposes. If, for the sake of convenience,

we compute the entire resultant work in terms of its dynamic effect, the round figure obtained is 90 million horse-power.* Allowing as equivalent of each horse-power the work of six strong men, the quantity is represented by 540 million man-power, continually active during 12 hours of each working day. This vast amount of work we 250 million Atlanticists alone have furnished to mankind—for the other 1250 million naturists have contributed nothing toward it—and all of this through the agency of the manganistic principle. Were we to assume that every tenth man of the 1250 million performed each day such hard and continuous work as that on which we have based our figuring, an assumption which is probably far too liberal, it would yield a product of not over 125 million man-power. We Atlanticists, the one-sixth portion of our earth's inhabitants, consequently accomplish with our manganistic work far more than four times as much as the others possibly can. The preponderance of manganists over naturists is not casual, therefore, but has been earned and paid for in useful work, and attains thereby, in a purely material sense, its justification. And all the more is this true, because much of our work is conveyed in its products to the naturist, serving so—when we consider the broad traits of the procedure, and not the incidents of still existing imperfections—for the dissemination and growth of civilization and culture. Thus it is that technical science becomes the propagator of general culture, a vigorous and indefatigable worker for the melioration and advancement of the human race.

This inherent worth, this importance of the technical sciences in cultural development, is of itself sufficient natural cause for the wide field which they have opened to education. In proceeding to investigate the subject of technical instruction we recognize at the very outset that in its highest grade, that of the polytechnic school,† it is not conceivable without a thorough study of the sciences. For manganistic technics must, in all of its operations, apply the forces of nature in exact conformity with their laws. The instruction in the polytechnic school has of necessity, therefore, to adopt as fundamental principles the three natural sciences—mechanics,

* On a basis of coal consumption of 1¼ kg. per horse-power per hour, and 12 hours of work per day, *i.e.*, 4½ tons of coal per horse-power per annum. According to statistical count and estimates, the energy of 20 million tons is actually reproduced in dynamic form.

† *Technische Hochschule.*

physics, and chemistry, and the all-measuring master art of mathematics. To these the separate technical courses are allied, the special studies in each depending for their title to a place in the curriculum upon the provision that every legitimate problem which may present itself in practice shall be approached from the highest standpoint of intelligence. The polytechnic schools, arising as they did out of the requirements of a gradually developing system of manganistic technics, date their existence from a much later period than the universities: it is only the present century that has substantially matured them, and even this not yet completely, for the signs of unrest which they have exhibited for some time past are indicative of a development that is still in progress.

It is worthy of remark here, that notwithstanding their intended pursuit of a strictly scientific aim, the technical schools have not yet concluded their peace with the universities. Even with the best of good will none of our efforts toward a real amalgamation of the two has ever been successful. In this connection I can vividly recall a bright, festal oration of the late Professor Kœchly, in which he dwelt on the social communion of the Zurich Polytechnic School and the University beneath a common roof. It was the same Kœchly who with Rüstow pursued technical philology, translating Philo of Byzanz, and Hero, and Aneas on the defence of cities, and Cæsar, and other works in which technical questions were always prominent. He said of the relations of the two schools: "And if not side by side, at least we can fight back to back." And to-day, after a lapse of twenty-four years, the fight is still carried on there, as in so many other places, back to back. It cannot be advanced as adequate explanation that the disparity in age is too great, or that the universities have let the right season slip by when the necessary polytechnic faculties might have been grafted to the older four. Deeper causes than these must militate against the coalescence. I look upon them as residing in the inherent difference in the purposes of instruction.

University education strives in all of its departments to enlarge the scope of human knowledge; the university pursues, to express it in brief, *the sciences of research.** In theology, in jurisprudence, in medicine, in all departments of the philosophic faculty, the advancement of pure knowledge is the one object of its endeavor.

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* *Die Wissenschaften des Erkennens.*

How different is the purpose at which technical instruction aims: it would teach and fit us to form and to fashion, to create what is new, to promote the objects of society by the application of science to the control of natural processes. The polytechnic school, in a word, promotes *the sc ences of accomplishment.** This contrast is decisive. It determines the whole tenor of instruction. In the university it carries its disciple ever farther toward specialization of study, toward segregation from the common matrix, and complete devotion to one object of research.† In the polytechnic school, on the other hand, the higher the student rises upon the foundation of firm and well ordered scientific elements, the greater becomes the necessity for coördination, for the organic, functional working of each separate study, and particularly of the disciplinary ones, and for an apt accommodation of the student to various related lines of activity. Later in life it is not otherwise, for every technical career may impose participation in organized operations, where on one man alone devolves the onerous responsibility for harmony of action in a complex whole; while university education fosters singleness of aim, and ever after, in a professional career, tends towards its further development.

It scarce may be necessary to mention that the limits between the two intellectual movements are not sharply defined, and particularly is this the case in the sphere of instruction itself. Not unfrequently, for example, the chemistry of the university, yielding to allurements, has swerved from its own course toward the field of active accomplishment; nor have the teachers of the polytechnic school always kept within the confines of a strictly technical treatment of their subjects. Hence the exchanges, both back and forth, in the forces of instructors, until a fair equilibrium is established. But besides, the true principles that should govern the polytechnic schools have not as yet crystallized everywhere, nor has the necessity for building up by thoroughly scientific methods received as unqualified an acknowledgment as it certainly should. Enlightened views in this direction, however, are steadily gaining ground, and we may regard the end as now being distinctly recognized.

Separately, therefore, as marching columns, the two forces are

* *Die Wissenschaften des Schaffens.*

† "Gentlemen," I have heard Dove say in one of his lectures on physics, "fix betimes upon a specialty."

advancing; their ranks, their aims are different; but in the same scientific plane they move, and always shall move, neighborly enough to extend to each other a friendly hand, and each striving to satisfy the full measure of all just demands. Let them, then, march on in peace; notwithstanding their separation both will surely do justice to their undertakings.

A blending of the two movements, be it said, has actually been tried; that is, it exists to-day in the United States. The "university" in America is both classic and technic. The experiences hitherto gathered, however, have not shown, so far as my observation allows me to judge, that the union can be permanently maintained, or that it has furthered the interests of education in the way that legislation had anticipated.

There is one among the polytechnic departments, that of architecture, concerning which doubts as to the aptness of our observations may arise. Many would sever the architectural course from the technical schools and incorporate it with the academies of fine arts. The advantages of such a change appear to me extremely doubtful, but that there is a certain inherent justification for the desire must be allowed. For architecture is, and always has been, an own, peculiar growth of naturism, which had flowered in surpassing beauty long ere the seeds of manganism germinated in the soil. We can understand, therefore, and must respect the tendency that has recently manifested itself in efforts to reintroduce studio instruction in the architectural department of the technical school. In my opinion, however, it is a matter of serious doubt whether the expected advantages will be realized. For it is simply the old naturistic method which is thereby revived; the personal excellence of the individual artist again becomes the object to be taught and transmitted. But meanwhile the close attachment of architecture to the polytechnic school has certainly inculcated in it one idea which unites it in essence with the school, and that is the setting of scientific laws at the very foundation—not laws of natural science, as in the other technical branches, but laws of æsthetics and of style, which become the highest intellectual guides in artistic creation, even as the laws of nature are the fundamental elements in the other departments. Of perhaps secondary, though not to be underestimated, importance for the connection of architecture with the technical schools is the circumstance that at the present day, and for our conditions of life, no building of any magnitude is

conceivable in which manganistic technics do not figure in as prominent a rôle as artistic composition; in problems of heating, lighting, ventilation, water supply, drainage and many others, it is a vital requisite, and accordingly demands the fitting studies.

Let us terminate our discussion by just glancing at the intermediate and lower technical schools, the industrial art schools included, as almost all of them are related to the architectural branch of the technical high school, or range within a dependent province of the academies of fine arts. The further down the scale we look the more definitely must we recognize in the humbler institutions the training schools for the actual performers, while the polytechnic high school undertakes to educate the designing and directing staff. It is a facile deduction from our doctrine to see that the lower in the system we go, the more must the acquirement of law cede to the learning of rule. In the lowest grade, instruction should in my opinion embrace only the rule, which of course must be an emanation of law, in complete harmony with it, if it is to maintain any claim to value. The teacher, therefore, must, or rather should, know the law, but he should abstain from teaching it. For the free and unconstrained view of the immediate performer is sure to be disturbed if he is beating his brains about the deeper law; it troubles him and disquiets him, for, as a learner in this grade, he still stands as a general thing upon the naturistic level. He is the soldier of technics. But just because of this the thorough assimilation of the rule becomes a source of strength and capacity. The higher upward in the scale, the more of law may be introduced, and the more may we gradually pass over to manganism. "Rule, example, law,"* thus perhaps the rising gradation might be expressed in concisest form.

It should be the important duty of boards of education everywhere to determine for the course of studies in each of the several grades of schools just how much of the law shall be added to, shall be shown as rational basis for, the rule. Quite natural as this may appear, yet from my own experience it will not be superfluous to direct attention to this point, because teachers, and especially the younger ones, too often err in failing to impose upon themselves a wise restriction, obstructing thus the pathway to their own success, and frequently thwarting the intentions of educators.

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* *Regel, Vorbild, Gesetz.*

BOOK REVIEWS.

MECHANICS OF MATERIALS. By Professor Mansfield Merriman. JOHN WILEY & SONS, Publishers. Price, $2.00.

In the process of designing an engineering construction the engineer is ultimately brought to face the problem of proportioning the various beams, ties, etc., of which his structure is composed. He first deals with his bridge as a unit, but later must determine the size of every strut, tie, and pin. This immediately opens to him a field of investigation where molecules are to be dealt with instead of the members of a framed structure; where, instead of applying the principle of the revolution of forces to finite length, definite in direction, he must discuss the stresses and resulting strains in fibres infinitely close in contact in all directions.

This distinct branch of statics Professor Merriman has most happily treated in his *Mechanics of Materials* recently brought before the public. In a work of about one hundred and fifty pages he has unfolded the principles of the subject in a remarkably lucid manner. As stated in the preface, it is intended as a text-book for technical schools and colleges, and any one who has had practical experience in instructing engineering students can see that it is eminently adapted for the purpose.

There are a number of excellent works on strains, etc., which fail in just this particular, of giving to the average student a clear, definite idea of what he is studying. After having more or less vaguely waded through them, he is, perhaps, in a condition to commence again and see the true significance of many important points that previously had no meaning to him. The difficulty is in the author's inability to place himself in the position of the student. He is unconsciously writing for the existing, rather than the would be engineer, and often does not realize that facts perfectly plain to him may be unknown to the novice. The author of *Mechanics of Materials* is well fitted by a long experience in instruction to appreciate this, and the result is a compactness and definiteness of statement that it has not been the privilege of the critic to enjoy in most text-books of this class. The aim has been to "form at first a definite nucleus in the mind of the student, around which may be later grouped the multitude of facts necessary in his own department of study and work."

In the first chapter stresses and strains are defined, and a clear idea given of working stress as distinguished from ultimate strength. At the present day, when engineers are beginning to base their proportioning on tests of the elastic limit of materials, it is interesting to note that Professor Merriman adheres to the method of making the working stress a function of the ultimate strength, because the latter can be as yet determined with so much greater definiteness, although the assigning of it with reference to the elastic limit is, in his opinion, much more rational, and may in time become the more important method. Chapter two treats of stresses in pipes, cylinders, and riveted joints. In the third and fourth chapters the stresses in beams are discussed. The usual formula for transverse stress is deduced, then that for deflection, and an interesting comparison is made of the relative strength of beams and cantilevers under various loads. For restrained and continuous beams the theorem of three movements is used, and the treatment is such as to clearly show their relations to simple supported beams.

Chapter five gives an admirable discussion of the empirical formulæ

for columns. Euler's, Hodgkinson's, Tredgold's, and Gordon's are successively given, and their relations explained. Gordon's formula is used in the subsequent pages, because it most satisfactorily represents the results of experiments, and also involves as a factor the maximum unit stress of the material. The torsion of shafts is discussed in chapter six. Chapter seven explains how combined stresses are to be treated, as when, for example, a beam is subjected to direct compression and transverse stress at the same time.

The last chapter of the book is devoted to several topics, one being an intimate investigation into the internal stresses of beams, beyond what is covered by the theory of flexure used in deducing the usual formulæ for them. This is of theoretical interest, but not necessary for a practical proportioning of beams, as the ordinary formulæ for transverse strain, although founded on approximate laws, suffice for all practical requirements. The discussion is, however, of great value to the student in aiding him to a clear idea of the limits of the theory of flexure, and of the manner in which the strains are distributed through the fibres. The results of Wöhler's experiments on the fatigue of materials, and the influence of its laws on the selection of working stresses are defined, and finally, tables of average ultimate strength, limits of elasticity, etc., for various materials, such as would be useful in working out problems in design are given.

The division into chapters and articles, each with an appropriate heading, is followed throughout, and numerous problems pertinent to the text are distributed through it. When the student has thoroughly mastered this book he can hardly fail to be familiar with the distinct and important branch of designing that it is intended to cover, namely, the proportioning of the parts to resist the stresses in the fibres of the materials used. J. L. G.

JAHRES-BERICHT UBER DIE LEISTUNGEN DER CHEMISCHEN TECHNOLOGIE, für das Jahr 1884. Fortgesetzt von Dr. Ferdinand Fischer, 414 Abbildungen. Verlag von OTTO WIGAND: Leipzig. 1885. 1360 pp.

This important work, now in its thirtieth year, is of value to all who follow any branch of chemical technology. One-fifth of the space is devoted to general metallurgy, of which the greater portion treats of iron and steel. Under the heading of Gold we find a reference to Browning's article on Gold Chlorination in California, which first appeared in the QUARTERLY, and in various places throughout the book we come across familiar names. The subject of food-stuffs and beverages occupies 360 pages, that of inorganic chemical manufactures 183 pages, organic products 164 pages, of which by far the greatest amount is occupied by the coal tar and other colors. The rest of the space is nearly evenly divided under the heads of glass, clay, cement, and artificial stone, the technology of textile fabrics, the technology of other organic products, including fats, glycerine, resins, tannin, glue, fertilizers, disinfectants, and that of fuels and electricity. Where the whole book is so carefully compiled it is impossible to select any one portion for special praise. We have not the least hesitation in saying that anybody interested in these subjects will find this book of very great value to them, and will never regret its cost. In order that future volumes may be made still more complete the editor requests that any papers treating of technological subjects in any way may be kindly forwarded to him. †

FRICTION AND LOST WORK IN MACHINERY AND MILLWORK. By Prof. Robert H. Thurston. JOHN WILEY & SONS: New York. 1885. $2.00.

The author has brought together in the 360 pages of this book the results of a long series of investigations into the facts and laws involved in the waste of energy by friction in machinery and millwork. The impossibility of reconciling the facts of common experience with the data supplied in engineering literature, is well known to all who have had any practice in designing machinery. The enormous variations in the actual values of the coefficients of friction, as produced by changes of pressure, of speed, and of temperature, have been shown in a thoroughly practical way, and a large number of relations have been established between the conditioning causes. The present volume contains a great deal which appeared under the title of *Friction and Lubrication* in 1879, but is much more extensive, and is brought up to date. Beside his own investigations Prof. Thurston has been able to avail himself of a great deal of experience acquired in his mechanical laboratory by students working under his direction, thus adding greatly to the value of the work.

The book begins very properly with a chapter on the theory of machinery. The second chapter defines friction, and treats the subject from the standpoint of mechanics, and collects in a very convenient form a number of the formulæ of dynamics of machinery, showing how friction militates against the efficiency of different mechanical devices. The discussion of journal and pivot friction is particularly full, bringing out very clearly the effect of different shapes and of different methods of fitting. The third and fourth chapters bring together, in a form particularly valuable to engineers, the information about the oils and greases which has heretofore only been accessible where a chemical library could be consulted. A few pages are also given to the methods of applying lubricants on bearing surfaces.

In the fifth chapter the Professor has, in conjunction with Prof. A. R. Leeds, brought together an admirable statement of the information in literature on the chemical and physical tests of oils. He discusses the methods of examination of oils to determine adulteration, density, viscosity, tendency to gum, acidity, endurance, cooling power, determination of the effects of temperature, and measurement of the coefficient of friction. The investigations of Calvert, Prescott, Cailletet, Chateau, Wurtz, Gerhardt, and Chancel, are referred to, and Chateau's tables of the color reactions of the oils are given with great fulness. Chandler's and Wiechmann's tables are quoted as standards for determination of density of oil in Baumé's degrees, and considerable space is given to a discussion of the oleographic or cohesion-figures.

The sixth chapter gives values of the coefficients for the different frictions as determined by different experimenters under a great variety of conditions. There are a number of tables and formulæ covering these investigations. The different forms of testing machines are illustrated and described, and the method of making tests with such apparatus is made very clear.

It is particularly, however, in the discussion of the friction of lubricated surfaces that the book presents its most signal value. The results of a large number of original experiments, and the deductions and conclusions from them, will not be found anywhere else in literature, and are

results. But look to other features which impart a character to our abodes. There is the gaslight in each house, and on the street, and in the public buildings. We owe it to a chemical storing element* of the fourth order (fire, retort, gasometer, conduit and cocks, neglecting all the secondary parts), and all of this wonderfully and systematically ramified through the town. The water supply for household and public use, when derived from a river, furnishes us with a driving element† of at least the sixth order. On our railways we are conveyed by driving elements of high order, whose service we regulate by means of others equally high; our freight is moved from place to place, from land to land, from continent to continent, in quantities greater a thousandfold than man and animals could carry. With a physical driving element we conduct a written and spoken message service the wide world over. And how do we proceed in war? In millions of chemical storing elements, great and small, and almost all of them of an advanced order, we carry potential energy out upon the distant battle-field, and there release it by means of driving elements of high combination. Out upon the broad ocean, hundreds of miles from land, the energy of a storing element, which we derive from a gift of nature, operates through driving elements of high order to carry us for weeks and months through wind and wave.

Highly potentialized storing elements, like coal, have been eagerly sought out in nature—at an early day naturistic man found the water-course, the subordinate element of transmission for the greater storing one in the mountain heights—and in future we may still find others, as petroleum was found, rediscovered we may say, three decades ago. In this nature provided us with a highly energized chemical storing element, particularly adapted to disengagement when its constituent particles were at a bright red heat. It really consisted of a number of storing elements, in some of which the engagement was so slight that their release was apt to occur unintentionally. We were obliged, therefore, to subject the natural product to a process of separation, which, by the manganistic means of fractional distillation, removed the groups of particles that were too unstable, and then only did we obtain a storing element that could be safely transported and come into general use. The restrictive police ordinances required, as we see, that the engagement should be a firm one, and how favorable

* *Spannwerk.* † *Treibwerk.*

in their action these regulations have proved, the results to-day abundantly testify. This liquid storing element which was obtained, so to say, almost ready made from nature, replaced for illuminating purposes a number of others which we had derived from the organic world with the aid of considerable manganistic machinery.

Let us, before concluding, direct our attention to another point. In a conflagration we see an unintentional release of chemical storing elements—the class to which so many bodies prove to belong. The ratchet has been raised against our will; with increasing velocity, frequently at raging speed, the tremendous storing element runs down. But we hasten to check its course with another driving element, once worked by man-power alone, but now just as commonly operated by a chemical storing element (fire and boiler) through the intervention of a mechanical driving element of high order. Sometimes, too, we apply the action of a chemical storing element directly upon the quenching water, as when we use a gas fire-engine, or chemical engine, as the Americans are wont to call it. In this latter case, the driving element for the water is of a much lower order than before. It furnishes an illustration of how driving elements, whose ultimate effects are the same, compete with one another by rival endeavors to reduce the magnitude of their ordinal coefficients, or, in other words, to diminish the number of simple elements in their combinations. Everywhere, therefore, the manganistic thought, the manganistic principle, with which we sustain so largely, and ease, and defend our lives, but with which, too, we accomplish so often the destruction of our fellows.

And, finally, our industries, the producers of all our articles of daily use, and in turn again, of the manganistic machinery we require—what powerful influences have they not exerted through manganism in promoting our general culture! Just here let us pause, to enter the subject a little more in detail, and fix, if possible, upon a unit or standard of measure.

An essential factor for manganistic work is coal. It is now mined to the extent of about 400 million tons per annum, and is used mainly for industrial purposes. The amount produced in excess of this quantity suffices to cover the consumption for general heating. We have, therefore, for each of the 300 working days in the year, 1⅓ million tons of coal which are applied to chemical, mechanical, and physico-technical purposes. If, for the sake of convenience,

we compute the entire resultant work in terms of its dynamic effect, the round figure obtained is 90 million horse-power.* Allowing as equivalent of each horse-power the work of six strong men, the quantity is represented by 540 million man-power, continually active during 12 hours of each working day. This vast amount of work we 250 million Atlanticists alone have furnished to mankind—for the other 1250 million naturists have contributed nothing toward it—and all of this through the agency of the manganistic principle. Were we to assume that every tenth man of the 1250 million performed each day such hard and continuous work as that on which we have based our figuring, an assumption which is probably far too liberal, it would yield a product of not over 125 million man-power. We Atlanticists, the one-sixth portion of our earth's inhabitants, consequently accomplish with our manganistic work far more than four times as much as the others possibly can. The preponderance of manganists over naturists is not casual, therefore, but has been earned and paid for in useful work, and attains thereby, in a purely material sense, its justification. And all the more is this true, because much of our work is conveyed in its products to the naturist, serving so—when we consider the broad traits of the procedure, and not the incidents of still existing imperfections—for the dissemination and growth of civilization and culture. Thus it is that technical science becomes the propagator of general culture, a vigorous and indefatigable worker for the melioration and advancement of the human race.

This inherent worth, this importance of the technical sciences in cultural development, is of itself sufficient natural cause for the wide field which they have opened to education. In proceeding to investigate the subject of technical instruction we recognize at the very outset that in its highest grade, that of the polytechnic school,† it is not conceivable without a thorough study of the sciences. For manganistic technics must, in all of its operations, apply the forces of nature in exact conformity with their laws. The instruction in the polytechnic school has of necessity, therefore, to adopt as fundamental principles the three natural sciences—mechanics,

* On a basis of coal consumption of 1¼ kg. per horse-power per hour, and 12 hours of work per day, *i.e.*, 4½ tons of coal per horse-power per annum. According to statistical count and estimates, the energy of 20 million tons is actually reproduced in dynamic form.

† *Technische Hochschule.*

physics, and chemistry, and the all-measuring master art of mathematics. To these the separate technical courses are allied, the special studies in each depending for their title to a place in the curriculum upon the provision that every legitimate problem which may present itself in practice shall be approached from the highest standpoint of intelligence. The polytechnic schools, arising as they did out of the requirements of a gradually developing system of manganistic technics, date their existence from a much later period than the universities: it is only the present century that has substantially matured them, and even this not yet completely, for the signs of unrest which they have exhibited for some time past are indicative of a development that is still in progress.

It is worthy of remark here, that notwithstanding their intended pursuit of a strictly scientific aim, the technical schools have not yet concluded their peace with the universities. Even with the best of good will none of our efforts toward a real amalgamation of the two has ever been successful. In this connection I can vividly recall a bright, festal oration of the late Professor Kœchly, in which he dwelt on the social communion of the Zurich Polytechnic School and the University beneath a common roof. It was the same Kœchly who with Rüstow pursued technical philology, translating Philo of Byzanz, and Hero, and Aneas on the defence of cities, and Cæsar, and other works in which technical questions were always prominent. He said of the relations of the two schools: "And if not side by side, at least we can fight back to back." And to-day, after a lapse of twenty-four years, the fight is still carried on there, as in so many other places, back to back. It cannot be advanced as adequate explanation that the disparity in age is too great, or that the universities have let the right season slip by when the necessary polytechnic faculties might have been grafted to the older four. Deeper causes than these must militate against the coalescence. I look upon them as residing in the inherent difference in the purposes of instruction.

University education strives in all of its departments to enlarge the scope of human knowledge; the university pursues, to express it in brief, *the sciences of research.** In theology, in jurisprudence, in medicine, in all departments of the philosophic faculty, the advancement of pure knowledge is the one object of its endeavor.

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* *Die Wissenschaften des Erkennens.*

How different is the purpose at which technical instruction aims: it would teach and fit us to form and to fashion, to create what is new, to promote the objects of society by the application of science to the control of natural processes. The polytechnic school, in a word, promotes *the sciences of accomplishment.** This contrast is decisive. It determines the whole tenor of instruction. In the university it carries its disciple ever farther toward specialization of study, toward segregation from the common matrix, and complete devotion to one object of research.† In the polytechnic school, on the other hand, the higher the student rises upon the foundation of firm and well ordered scientific elements, the greater becomes the necessity for coördination, for the organic, functional working of each separate study, and particularly of the disciplinary ones, and for an apt accommodation of the student to various related lines of activity. Later in life it is not otherwise, for every technical career may impose participation in organized operations, where on one man alone devolves the onerous responsibility for harmony of action in a complex whole; while university education fosters singleness of aim, and ever after, in a professional career, tends towards its further development.

It scarce may be necessary to mention that the limits between the two intellectual movements are not sharply defined, and particularly is this the case in the sphere of instruction itself. Not unfrequently, for example, the chemistry of the university, yielding to allurements, has swerved from its own course toward the field of active accomplishment; nor have the teachers of the polytechnic school always kept within the confines of a strictly technical treatment of their subjects. Hence the exchanges, both back and forth, in the forces of instructors, until a fair equilibrium is established. But besides, the true principles that should govern the polytechnic schools have not as yet crystallized everywhere, nor has the necessity for building up by thoroughly scientific methods received as unqualified an acknowledgment as it certainly should. Enlightened views in this direction, however, are steadily gaining ground, and we may regard the end as now being distinctly recognized.

Separately, therefore, as marching columns, the two forces are

* *Die Wissenschaften des Schaffens.*

† "Gentlemen," I have heard Dove say in one of his lectures on physics, "fix betimes upon a specialty."

advancing; their ranks, their aims are different; but in the same scientific plane they move, and always shall move, neighborly enough to extend to each other a friendly hand, and each striving to satisfy the full measure of all just demands. Let them, then, march on in peace; notwithstanding their separation both will surely do justice to their undertakings.

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There is one among the polytechnic departments, that of architecture, concerning which doubts as to the aptness of our observations may arise. Many would sever the architectural course from the technical schools and incorporate it with the academies of fine arts. The advantages of such a change appear to me extremely doubtful, but that there is a certain inherent justification for the desire must be allowed. For architecture is, and always has been, an own, peculiar growth of naturism, which had flowered in surpassing beauty long ere the seeds of manganism germinated in the soil. We can understand, therefore, and must respect the tendency that has recently manifested itself in efforts to reintroduce studio instruction in the architectural department of the technical school. In my opinion, however, it is a matter of serious doubt whether the expected advantages will be realized. For it is simply the old naturistic method which is thereby revived; the personal excellence of the individual artist again becomes the object to be taught and transmitted. But meanwhile the close attachment of architecture to the polytechnic school has certainly inculcated in it one idea which unites it in essence with the school, and that is the setting of scientific laws at the very foundation—not laws of natural science, as in the other technical branches, but laws of æsthetics and of style, which become the highest intellectual guides in artistic creation, even as the laws of nature are the fundamental elements in the other departments. Of perhaps secondary, though not to be underestimated, importance for the connection of architecture with the technical schools is the circumstance that at the present day, and for our conditions of life, no building of any magnitude is

deserving of careful study. The work concludes with an interesting chapter on the valuation of lubricants, from the standpoint of the cost of power lost in friction. The author shows that the cost per gallon of a machinery-oil is not the real index of its value, but that a cheaper oil which admits of a little greater friction of bearings may cause so much greater an expenditure of fuel, in overcoming increased resistances, as far to outweigh the supposed saving in first cost. This subject, with practical illustrations of the working of the principle, has been even more fully worked out in a paper presented in the sixth volume of the *Transactions of Mechanical Engineers*, to which the reader may be referred.

The book contains an index and table of contents, and is dedicated to Mr. C. A. Hirn. F. R. H.

ON SECONDARY ENLARGEMENTS OF MINERAL FRAGMENTS IN CERTAIN ROCKS. By R. D. Irving and C. R. Van Hise.*

This valuable contribution to the literature of the microscopic structure of fragmental rocks, presents a multitude of interesting facts and observations, from which conclusions of great importance are drawn, relative to the genesis as well as the intimate structure of quartzites, to which the authors' studies have mainly been directed. It has long been taught that true quartzites are derived from pre-existing sandstones, the process including the rearrangement and recrystallization of the component quartz grains. In the present paper it is conclusively shown that the principal cause of induration and hardening is the deposition of silica between the individual grains, which at the same time are enlarged, the accrement being in perfect optical continuity with the original fragments. The grains are thus firmly cemented together and rigidly interlocked. The studies were principally made on Huronian quartzites of the Northwest, but indurated sandstones and quartzites of more recent geologic age have been incidentally examined, and with the same general results.

Mr. Van Hise contributes the chapter on "Enlargements of Feldspar Fragments in certain Keeweenawan Sandstones."

There seems to have been held, by some, an erroneous idea relative to the minute structure of sandstones, for, not long since, the writer was present at a meeting of a learned society, where a prominent member expressed his disbelief in the importance of any cementing material in sandstones. Professor Irving's paper will also prove an obstacle to those who have adopted the hypothesis of a chemical origin for all crystalline rocks.

The paper is beautifully illustrated with colored lithographs of crystal-face enlargements, and of thin sections of quartzites and sandstones.

N. L. B.

* Bull. U. S. Geol. Survey, No. 8, 1884.

ABSTRACT.

MINING.

The Banca Tin Mines.

The cassiterite of Banca occurs: in thin, irregular veins in granite; in fissures in the adjoining sandstones and quartzites; in the atmospherically disintegrated rocks which everywhere cover the granite; and in the old river-valleys. The valleys are cut in the granite, their beds are of compact clay in which quartz or mica predominates, according to the nature of the adjacent rocks. The strata carrying the mineral rest upon the clay bed, and have evidently undergone a species of natural concentration, the feldspars being changed to clay and carried away, and the cassiterite deposited, the larger grains near the valley-origin, and the smaller further and further down stream as their size diminishes. Above the mineral-bearing strata are alternations of clay and sand, and finally a black clayey or sandy soil in which the modern river flows.

The total thickness of the deposit rarely exceeds ten metres, and the strata containing the tin average $\frac{3}{10}$–$\frac{6}{10}$ metre.

The minerals associated with the cassiterite are quartz, clay, tourmaline, mica, manganese minerals, sometimes a little gold pyrites, and menaccanite, and, in one deposit only, wolframite.

The deposits have been very carefully mapped, and the topographical work has been accompanied by soundings, from which the values of the deposits, per square metre of deposit and per cubic metre of earth removed, have been deduced.

The soundings have proved that the valleys are very far from being equally rich, that there is little chance of success in the valleys not yet exploited, and that the moment is not far distant when it will be necessary to mine the *stock-werks* which gave birth to the alluvial deposits.

The mining is entirely done by Chinese, and is divided into annual tasks, because this permits the most advantageous use of the rainy season for washing the mineral.

The men, in companies proportioned to the allotted year's task, commence work immediately after the Chinese New Year (February). A dike is constructed up the river, and the water is carried along the flanks of the valley by two canals. A third canal is dug as deep as possible to remove the waste. When the deposit has drained dry, a mark is made on the surface to show the intended limits of the year's work, the vegetation is burned, and the running water brought to soften the earth, and facilitate its removal into the excavation of the preceding year.

When the first metre of soil has been removed in this way, the water is confined to wooden sluices, and a dike built between the two excavations; the earth is then raised in baskets, thrown into the sluices, and carried over the dike by the water.

After the removal of the three upper metres, *over the entire year's allotment,* the deposit is divided into small portions which are in succession exploited from top to bottom. The sterile earth is simply thrown back

into the nearest excavation until the mineral is reached. The mineral is carried to the surface in baskets suspended from each end of a shoulder staff, and is there sifted to separate the coarser gravel, and washed in rude sluices. The settlings are rewashed in troughs, and a short time before smelting are again washed.

The ore is reduced by charcoal without flux, on a rectangular hearth of quartzose clay, set against a slightly higher wall. The hearth proper is semicircular, and connects in front with an exterior basin, the tuyere is a little higher than the tap-hole, and the blast is furnished by a very primitive double-acting piston-bellows.

The mineral yields about seventy per cent. of tin in the fusion, and about three more units are obtained by retreating the scorias.

The yield for a night of smelting is about fifteen hundred kilos per furnace.

No refining is needed.

The mean average yield, between 1872 and 1881, has been:

Island of Banca, ...	4163 metric tons.
Island of Biliton, ...	4164 " "

Revue Universelle des Mines, January and February, 1885.

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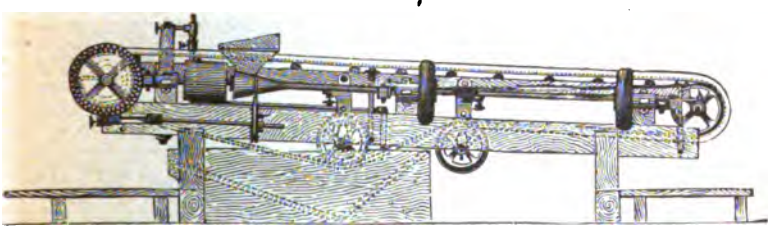

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WORKING PLACER DEPOSITS IN THE UNITED STATES.

BY T. EGLESTON, PH.D.

ALTHOUGH it is less than forty years since gold was discovered in California, it is already very difficult to ascertain what the truth of the history is. This is owing to the fact that long before the real discovery, gold which was found in Mexico, was credited to California, because it was brought to the East by vessels which reported it as coming from there. It is also owing in part to the fact that but little was known of the country, and that travellers who wrote about it, in the desire to feed the appetite for the marvellous, reported what they imagined rather than what they found, as coming from them. Others, from selfish or mercenary motives, reported what they hoped would bring emigration or raise the value of the lands they owned, so that little or no credit should be given to what was written previous to about the close of the last century. There does not seem to be any truth in the statement that Sir Francis Drake found gold in California (New Albion) in 1579. That found by Captain Jediah S. Smith during his overland journey in 1826 was found in the Sierra Nevada Mountains, and not in California. One of the first authentic records of finding gold was near the Colorado River, in San Diego County, in 1775. But little work was, however, done at this place, as the ground was too poor

to justify any serious exploitation of it.* The true history of the subject appears to be that towards the close of the year 1833,† or the beginning of 1834, J. P. L. Leese came to Los Angeles from Taos, New Mexico, bringing grain gold which had been washed out of the placers there. About the same time, a Spaniard named Palacios came to the same place from Guaymas, in the State of Sonora, bringing with him silver bullion in bars and bricks, and gold in grains from Sonora. Both lots together amounted to about ten thousand dollars in value, and were for the most part taken out of the country, being carried by trading ships to Boston. A part of it was worked up by the jewellers in Los Angeles, but no tradition or belief that any part of California contained gold in workable quantities can be authenticated up to about 1841. In the early part of that year, Don Andres Castillero, a Mexican gentleman of scientific education, went to Los Angeles and Santa Barbara, and saw what he believed to be evidences of the occurrence of gold near the rancho of Las Virgines, and brought with him specimens to Santa Barbara, which he said indicated placer gold. In the month of June, 1841, Francisco Lopez, who had met Castillero in Santa Barbara, while riding over the ranch of San Francisquito, about thirty-five miles north of Los Angeles, found gold in a field of wild onions, which he took back with him when he returned to Santa Barbara. The rumor soon spread, and although very few of the Californians had any practical knowledge of gold-washing processes, there were a number of natives of Sonora, and other parts of Mexico and California, especially in the central part of it, who had seen placers worked, and during the first two rainy seasons following, a few hundred adventurers, from San Diego to San Luis Obispo, flocked into that part of the country and were profitably engaged in winnowing and washing gold, which was sold at Los Angeles every year. During the first year Lopez is said to have taken out about $8000. The value of 18 ounces sent in November, 1842, to the U. S. Mint at Philadelphia, was said to be about $340.

It is an undoubted fact that the Jesuit missionaries, for many years previous to any reported discovery of it, had known that there was gold in the country, and placer deposits had even been worked in a small way at their station of San Fernando in Los An-

* Report of the State Mineralogist of California, 1884, p. 217.

† Eng. and Min. Jour., vol. xxxii., p. 170.

geles County, but it was discouraged by the priests, who, believing that the passion for gold was demoralizing to their people, taught them that the working for, and the possession of gold, was detrimental to their own interests, and contrary to the true principles of religion. The simple folk accepted the dictum, and but for the rumors which came to the East so repeatedly, and from such different sources that they could not be treated as myths, the gold might not have been discovered for many years.

A carpenter, named James W. Marshall, who had been attracted to California, and was working as a millwright putting up a saw-mill on what is now known as Sutter's mill, at Coloma, El Dorado Co., on January 19th, 1848,* made the discovery of gold in the sluice of the mill which he was building, and subsequently in other places. On the morning of this day, Marshall went out to superintend the men, and after closing the forebay gate, shutting off the water, walked down the tail-race to study the character of the sand and gravel which had been removed during the previous night, for he had conceived the idea that there were minerals in the hills, and he said so to Sutter, who had only laughed at him. As he was examining the débris, his eye caught the yellow metal lodged in a crevice on a natural riffle in soft granite, some six inches under the water. After a close examination, he suspected that this substance was gold. Being a shrewd man, who realized the importance of his discovery, he went on with his work as usual, showing the nugget to his men, however, and making some conjectures about the extent of the gold-fields, watching carefully for any further developments, and in the course of a few days had collected a number of ounces of the metal. About four days after the discovery, he had occasion to go to Sutter's Fort, now Sacramento; so taking the grains with him, he examined the rest of the country, and again discovered the same metal in a ravine in the foot-hills, and at a place known as Mormon Island. Sleeping under an oak tree, some ten miles from the fort, he reached there the next morning, and soon put an end to all doubt as to the value of the discovery that he had made. It has been claimed that the gold was shown to Marshall by Peter L. Weimer, one of his men. Both men appear to have been together when the nugget was found, but Marshall, having a horse, went to tell Sutter of his

* Min. Resources U. S., 1872, p. 111.

discovery. No one of the three, however, knew that what had been found was gold until it had been assayed at San Francisco.

These discoveries were soon noised abroad, and caused the abandonment of the poor placers of the southern part of the country, and the influx of a host which depopulated whole villages and towns, the city of San Francisco amongst them, and caused the wildest excitement all over the East.

Although the discovery was so memorable, and the results of it were so important, it is hardly possible to-day to distinguish the spot where it was made. It has been proposed many times that a monument should be erected at Coloma, on the site of Sutter's mill, but it has never yet been done, and it is not now likely that the spot can be distinguished with sufficient certainty to make it worth while to attempt to mark it, so that it is not probable that any monument will be erected.

Marshall died in Kelsey, California, on August 12th, 1885, in great poverty and distress. His cattle were killed and his claims jumped by men who hunted him during the last years of his life in the hope of discovering the secret of finding gold, which they believed him to possess. His life was made wretched by his disappointment in never finding the fortune he was seeking, and which he always supposed to be almost within his grasp, but which he never succeeded in obtaining. Some years before his death, the Legislature of California appropriated $200 a month as a pension for him for two years. At the end of this time the appropriation was not renewed, and the man who had been the means of the finding of one billion, six hundred million dollars' worth of treasure in that State, died a wretched pauper.

PLACER DEPOSITS.

When gold is found in sand or gravel, the deposits are called placers. These are formed by the accumulations of the débris washed down by ancient rivers, often forming deposits of considerable depth and variable richness, which have themselves sometimes been again torn up and redistributed by more modern streams, forming deposits of comparatively recent origin. We have thus the ancient and the modern placer. They have, also, sometimes been either wholly formed, or, in some cases, only partially, by the precipitation, of the gold dissolved in the waters of

filtration, by organic matter.* According to the depth at which the gold is found, they are called shallow or deep placers. The shallow placers are those deposits which extend to the depth of about ten feet or less from the surface. It was in these deposits that the gold was originally found in California, and as they are usually very rich, they are the most eagerly looked for in the early settlement of a country. They exist, scattered over a great extent of territory, and were so rich and so largely worked that in the first five years after their discovery, they produced, notwithstanding the wasteful and primitive methods which were used to work them, $180,000,000. They are not always of the same richness and value, nor are they always capable of being worked. Abundance of water is the chief requisite to their successful treatment, and even then they are often worked on a wrong theory, and only while the pay-dirt continues very rich, and by the rudest and most wasteful methods, much more being lost than was ever extracted. They were often abandoned as worked out before their real richness was discovered, some of them having been thus worked over two or three times in succession. Even now the Chinese make a good living by working over these old tails. Up to the time of the invention of hydraulic mining, the shallow placers furnished nearly the whole of the gold produced in California. They were abandoned, because with the primitive means in use at that day, they were no longer profitable, but they are by no means exhausted. It is not improbable that it will, in the near future, be found profitable to work them over again on a large scale, both because the gold was only partially extracted as well as because the gold has concentrated in them from precipitation of the minute quantities of gold contained in the waters of infiltration.

It is estimated† that since 1851 the total amount of gold produced in the world has been $4,500,000,000. Of this sum, California has produced about $1,125,000,000, and of this amount, $1,000,000,000 have been produced by the working of the auriferous gravels.

Not unfrequently small particles of the precious metal were found in the roots of the grass or of small shrubs torn up by accident or design. Sometimes the shining particles of gold were seen

* Trans. Am. Inst. Mining Eng., vol. ix., p. 643.

† Prod. Gold and Silver in U. S., 1881, p. 616.

in the streams whose course lay through the ancient deposits or received the drainage waters from them. It was not till long after these modes of occurrence of gold had become known that it was discovered that the precious metal might be found deep down in the earth on the bedrock of ancient rivers which had been covered with detritus to the depth of hundreds of feet. These modes of occurrence give rise to many methods of exploitation of gold, the principal of which are known as placer and vein mining. The placers are divided into shallow and deep placers, and the method of mining varies, according to the manner in which the gold occurs in each, as follows:

In Shallow Placers.

Gravel mining.
Sluice mining.
Beach mining.
River or bar mining.

Deep Placers.

Hydraulic mining, { Surface mining. / Drift mining.

Vein Mining.

Amalgamation, { Dry crushing. / Wet crushing.
Concentration and chlorination.

GRAVEL MINING.

While the shallow placers are near the surface, the deep placers, as the name implies, have sometimes their richest portions near the bedrock, which is often two to three hundred feet below the surface, where the ground is generally quite poor. The formation of both these deposits is owing to the same general causes. In many cases the shallow placers are formed by the destruction of pre-existing veins, and the accumulation of the gold from the detritus by the action of the waters. In others they have undoubtedly been formed by the infiltration of small quantities of gold in solution in water which has percolated* through them

* Trans. Am. Inst. Min. Eng., vol. ix., p. 644.

from the surface, and has been thrown down in the sand by organic matter, either dissolved in the water or scattered as solid particles through the sand.

The methods by which shallow placers are worked differ from each other more by reason of the differences in the occurrence of the gold than because of any difference in the principle of the methods employed to extract it. The shallow placers are frequently divided into wet and dry diggings, wet being those which are found in the bars and beds of streams, dry being those which are found in the gulleys and the bottoms of the streams which are large in some seasons of the year, and dry up at others, so that water can only be had during certain portions of the year. These last claims, where the water is transient, are sometimes called "gulch claims." They are also sometimes called "dry diggings," and are not unfrequently very rich in gold. All these were worked in the early days, but as they have ceased to be very profitable, they are now left to the attention of Chinamen.

In the very early days the pan was almost the only instrument used. It still holds its place as a prospecting tool, although it has lost its importance as a working instrument. At first, it was any household pan that could be made available. Now it is made of Russia sheet-iron, but as its shape is such as is convenient for household purposes, the same pan is used for one and the other. Upon its skilful manipulation the success of the miners depended, and they soon learned to use it so that in the rich deposits or even in the poorer ones, they handled it with almost artistic skill. The work performed by it was small in amount, but so certain in its results, that its revelations were regarded as infallible, and districts were overrun or abandoned according to the results which it gave. In studying the method of its use, the work may be divided into periods, which will generally be found to correspond to five different phases or sets of motions: *First.* The pan is filled with the pay-dirt to about two-thirds of its capacity; it is then, according as water is abundant or scarce, placed in the bottom of a shallow stream, or set in a hole filled with water. It is usually placed on the bottom so that the operator can use both his hands. Occasionally it is held in one hand while the work is done with the other. The whole contents are thoroughly stirred with one or both hands. Any lumps that may be found are broken and mashed between the fingers, so as to mix them all thoroughly with the water. When the

whole contents of the pan are in the condition of a thin homogeneous mud, the first part of the operation is finished. *Second.* The pan is now taken in both hands, a little behind its diameter, raised nearly to the level of the surface of the water, but a little below it, inclined away from the operator, so that a very slight movement disengages all the light matter and causes it to pass over the edge of the pan. This is continued until the water which passes off is perfectly clear. There will remain in the pan, after all the lumps of clay have been broken up, only pebbles, gravel, heavy sand, usually black, and gold-dust. The *third* stage commences with raising the pan above the water and picking out all the large pieces, examining them before they are thrown away to see that they do not contain gold. The pan contains but little water, so that it is quite easy to get rid of the large pieces of sterile matter, either picking them out or pushing them over the edge. There will be now in the pan little else than small-sized pebbles, the black sand, and the gold. The *fourth* stage is to incline the pan a little under the water, causing the next smaller-sized pieces to pass in such a thin stream over the edge of the pan that every particle is visible, so as to be sure that no gold goes over. The *fifth* stage consists in lifting the pan out of the water with a little water in it, and by a dexterous movement of the wrist, causing the sand to pass with the water across the bottom of the pan, leaving "the color" on the opposite side. The pan is then inclined to the side where the sand is, and this is washed out with small quantities of water taken up in the hollow of the hand and poured on it, causing it to pass out, leaving the gold behind. The black sand, however, still remains with the gold. There are three ways of getting rid of it. The first is with a magnet when the sand is magnetic; the second, by dexterously blowing away the sand after it has become dry, and the third by the use of mercury. The first two are the methods most generally used.

The ordinary miner's pan has a flat bottom; the Mexican and Brazilian modification of it is a round piece of wood or metal, which is conical, and not over three inches deep in the centre, and is called a *batea.* It requires skilful manipulation. On the principle of this implement some successful prospecting and working tools have been made.

Great use was made in the early days of the *horn spoon*, which

was cut out of a large ox-horn so as to make a sort of spoon about a foot long by two to three inches wide. Horns of dark color, or those which were black at one end at least, were much sought after for this purpose, as the gold is so much more readily seen on a dark than on a light surface.

All of these were rather prospecting tools than mining implements. They were, and still are, used in all new fields, both as prospecting and working tools while the pay-dirt is rich, but when it becomes poor, so that the quantity to be treated in a given time must be great, they are no longer capable of treating the large amount necessary to give a fair return for the labor, so that if water was no' abundant, the field was abandoned, or if it was, "machines" were substituted for the pan. The first of these used was the "cradle," or "rocker," as it is often called. This was a box on rockers, having a grating on the top 20 inches square, called the riddle, on which the gravel was placed. Water was poured on this, which carried the pay-dirt through. It fell on an inclined blanket-apron, which carried it to the back of the lower part. On the lower part are two riffle-bars, about one inch high. The cradle is slightly inclined to the front, so that all the dirt and water put in at the top ran down the apron to the back, over the bottom, and out at the front. The weight of the water used in the cradle is from three to four times the weight of the dirt to be treated, so that the work must be done near the water. The very fine gold is caught on the blanket-apron, the coarser particles behind the riffles; the larger stones are easily picked out from the riddle, and the small ones are left to help break up the tough dirt. The motion of the cradle must be constant to dissolve the dirt and keep it loose; otherwise it would pack hard and the gold would run over it. The amount of water used must be large, or the excess of dirt would not be carried off. When the blanket is to be cleaned up the riddle is removed and the blanket taken out and washed in a pail. All that is caught has to be panned. If the rocker is to be worked by one man, the water is collected in a hole made at his right hand. With the left he moves the rocker, and with the right he pours water over the dirt. If the "pay" is near the water it is brought in pails or wheelbarrows. If this cannot be done, it is "packed" on mules or brought in wagons. If worked by two men, one brings the water and the other rocks. If the water can be brought by a sluice two men can work rapidly. One

man can treat one and a half cubic yards in a day; two men can do nearly three times as much. When the gold is fine, the cradle, though very cheap, is not economical, as most of this gold is lost. When it is coarse and the supply of water uncertain, it is a very useful machine; but its output is very small, and the amount of labor required to be done with it is very large. At the best it can only treat half the amount that can be washed in a tom, with the same amount of labor. Its use has been abandoned in this country, except by the Chinese.

The rocker was replaced by the "tom," which was really only two launders or sluices, and required a supply of water from another sluice. The apparatus consists of an upper sluice-box about 14 feet long, 20 inches wide at the upper end and 30 inches at the lower, with an incline of a little less than one inch to the foot. The lower end of the launder is cut off at an angle of 45°. This is closed by a grating of punched sheet-iron, which allows everything but large stones to pass. This ends over a riffle-box, about 3 feet wide, and 11 to 12 feet long at the same incline, with five or more riffles, into which mercury is put. One man throws in the dirt, another stirs it, breaks up the lumps of clay and dirt, and removes the large stones. The gold is caught in the riffles, which are sometimes supplemented with blankets. It is worked by from two to four men. Two men will do five times as much with the tom as with the rocker. It is never worth while to use the tom except when the gold is in coarse grains and the quantity of pay dirt small. It is now only used by the Chinese, who use it to work over the "exhausted" tails of the abandoned workings of the early miners, and are quite satisfied if they only make seventy-five cents, or even less, a day.

Sluice Mining.

Almost all the shallow placer deposits are now worked by sluices. These sluices are tools of great value and simplicity, and their invention and use has probably done as much to increase the wealth of California as any other one thing. They were for many years the most important instruments used in working the placers. The sluice was invented in California, although it has been used elsewhere under, however, very different conditions. It has been brought to its greatest simplicity, as well as its greatest perfection in this country. It is made of rough unplaned boards, and is

of almost any length over 50 feet, and is not less than 8 inches wide and 9 inches deep. The width is usually 16 to 18 inches; it may be thousands of feet in length. The boards are an inch and a half thick and 12 feet long, just exactly as they come from the mill. The sluice is made in sections. Each section is called a box, and as the boards are from 12 to 14 feet long, the sluice is said to be made up of so many boxes, which indicates the length perfectly. The sides of these boxes are from 18 inches to 2 feet high. The bottom boards are sawed for this special purpose, and are 4 inches wider at the lower than at the upper end. The narrow end is intended to fit into the wide one of the upper end of the next one below. The sides are nailed upon the bottom. No attempt is made to make them tight, as the joints become filled with sand or swell tight after a little use. The sluices thus made rest upon trestles with a descent varying from 8 to 18 inches in a box. This descent is called its grade, and if the fall from one end of the box to the other be 8 inches, it is said to have an 8-inch grade. It is generally not desirable to have a grade of less than 8 or more than 20 inches. The boxes are put up on trestles with great ease and taken down as easily, and are transported from one claim to another after the placer has been worked out. The amount of water to be used in a sluice will depend on a great variety of circumstances, especially the quality and quantity of pay dirt to be treated in a given time. To work successfully there should be plenty of water. In general the depth of water on the bottom of the sluice should not be less than two inches.

The grade depends upon the character of the "pay," as the dirt containing the gold is called, the length of the sluice, and the character of the ground, and the quantity of water available. The steeper the grade the more rapidly the dirt will be worked, but the greater likelihood there will be of loss in gold. Ordinary dirt will be entirely broken up within from 200 to 250 feet of the point where it is charged, and from here on, the sluice has but one function, namely, to catch the gold. The point being determined at which a given quality of dirt will be completely broken up, the grade beyond that point is usually lowered, but must always be sufficient to carry off the excess of material. It is rare that a sluice is made long enough to catch more than 50 per cent. of the gold contained in the sand. Tough, clayey material requires a steeper grade than any other. To break up this tough clay the flow must be rapid,

but in such a case the sluice must be long. The grade must always be less in a short sluice than in a long one. This is owing to the fact that there is always danger that a rapid current over a short distance will carry off the gold, whereas in a longer distance it would probably be caught. When the amount to be treated is large, the grade should be high, the water abundant, as also when the ground contains a considerable quantity of large boulders. It is unusual for the sluice to have the same grade throughout. It is generally made steep at first, so as to break up any tough material, and less steep afterward. Usually, however, the miner makes the grade conform to the lay of the ground, whatever that may be, and regulates the height of the trestle by the ease with which he can throw the dirt into the sluice. There is always danger, when the dirt contains any plastic clay, unless the grade is very high, that this will roll over and over on the bottom, and take up the gold which has already been deposited and carry it out into the tails at the end of the sluice. Such balls may sometimes roll a thousand feet without breaking. For this reason special care is taken to make the sluice for this kind of work very steep at first to break up the clay, and lower afterwards to catch the gold, or, better still, no attempt is made to work the clay in the sluice; but it is first puddled either by hand or machine, and not introduced into the sluice until it is in the condition of liquid mud. For this reason when clay balls are found in the sluice they should at once be removed, broken up, and puddled. When there is but little clay hard work is all that is necessary to break up the clay in the sluice, but when there is much of it it is cheaper to puddle the whole than to run the risk of losing large quantities of gold. When there is a great deal of gravel or large stone to be removed, under currents are used. For this purpose the end box is left open, the bottom for the width of the sluice is covered with a grating sufficiently coarse to allow all the fine stuff to pass through. Under this a short sluice at right angles to the direction of the main sluice is placed, which ends over another sluice parallel to the main one but at a lower grade. The material coming from above, which is fine enough, passes the grating and goes on in the other sluice. The stones and coarse stuff fall out at the open end by the force of their own velocity.

The gold is caught in the lower end of the sluice by various devices, which consist of the insertion of some kind of false

bottom. Even though the boards are rough, with their splinters turned up stream, they would soon be worn smooth by the material passing over them, and the gold be swept away in the current. In order to prevent both the mercury and the material from running out of the box too quickly, narrow strips of board are fixed on the bottom. These may be either transverse or longitudinal, or both. These strips are called "riffles." Sometimes, where the works are to last for a considerable time, the sides of the sluice are protected with a "lining," so that they will not be too rapidly worn. The longitudinal riffles are from 2 to 4 inches thick, 3 to 7 inches wide and 6 feet long. From two to four are placed on the bottom of the box. Each box will, therefore, contain two such riffle bars, and whatever the sluice contains, whether two or more, is called a set. They are wedged to their places, and held together by a certain number of transverse riffles, so that they may be easily removed for a clean-up. It would not do to put them in permanently with nails, as they must be often taken out. Immediately after a clean-up the riffles are adjusted in the boxes. There will thus be in each box, with longitudinal riffles, sections which will be 6 feet long, and the length and depth between the riffle bars. When the sand is fine, transverse riffles are often used alone, but when there is a small amount of pasty material to be broken up, zig-zag riffles at an angle of 45° are sometimes used. These riffles reach nearly across the sluice and are nailed to the bottom. The water, with the fine metal, passes over them, while the coarse gold and the clay go from side to side with considerable velocity. Such riffles do not catch the gold, but only help to break up the clay. They are placed in the upper part of the sluice, which must always be terminated by the ordinary riffles. Occasionally the sluice has a wooden pavement, made of blocks cut across the grain of the wood and set several inches apart, but so that they can be easily removed. They are not so permanent as a rock pavement, and are not much used except in hydraulic mining.

The coarse gold is easily caught at the head of the sluice, but much of the fine gold would be lost if nothing farther was done to catch it. For this purpose mercury is put into the head of the sluice, being poured through a strainer to divide it into small streams or in some other way broken up into small globules. It is usually not added until from one to two hours after the commencement of the working. It is caught in the riffles and amal-

gamates the fine gold. Sometimes the pores of the riffle bars are impregnated with mercury by introducing a gas-pipe at one end, and filling it with mercury to a considerable height. This is very troublesome and scarcely pays for the work it occasions, as the riffles must be cleaned by scraping the amalgam formed from the wood. It gives as much trouble as copper plates, and is much less effective. Copper plates are not often used in placer working. These are extensively used in battery sluicing. When they are used they are amalgamated by cleaning them first with dilute nitric acid, which removes the dirt, and then rubbing them with mercuric nitrate or with mercury alone, which is rubbed on with a piece of cloth. When the former is used the nitric acid brightens the plates, and the mercury at once amalgamates with the copper; the green slime of basic copper nitrate thus formed must be carefully rubbed off. It is much quicker to clean the copper with potassium cyanide, but this requires more care, as if too strong a solution is used, or the plates are left in it for too long a time, they become black and will not amalgamate. When plates are used the current must be slow and the water shallow. A swift current carries off the fine gold, and deep water does not give the gold time to settle. When a speck of gold settles on any part of the plate others will collect around it. The more gold there is on the plate the better it catches the gold, so that when the miners can afford it they amalgamate the plates with gold; when they cannot afford that, with silver, and where that cannot be done with mercury alone. The amalgam is left to accumulate until it is from ¼ to ⅜ of an inch thick, but not thicker, as there is danger of its being stolen. It is then in a hard mass, which must be separated by force. To do this the plate is heated until it is a little hotter than the hand can bear, and the amalgam separated by scraping or with a chisel. A copper plate $\frac{1}{10}$th of an inch thick, after it is once amalgamated, can be made to last from five to ten years if treated with care. An amalgamated copper plate is better than a bed of mercury of the same size, but it must be kept clean by rubbing. This is usually done with an india-rubber tool, not unlike the one so widely used for cleaning windows. The time between two clean-ups is called a "run," which usually lasts from six to ten days. The sluice is generally run only in the day, but is sometimes worked day and night. Men must be kept on the watch to see that the material coming down does not choke the

sluice so as to make the water pour over the sides. Whenever a dam commences to form they break it up with an iron bar and throw out the stones with a fork.

Sometimes the sluices are made double by running a partition through the centre of a large one. This is done either when two companies are working on the same ground, in order to avoid the extra cost of two separate sluices, or when, because of the variable supply of water, it is desirable to run double the amount while the water lasts.

Sometimes, in order to prevent unlawful clean-ups at night, the sluices are paved with stone. These paved sluices, however, are not usual except in hydraulic mining. An active man can pave in a day eight hundred square feet of such sluice, made of cobble stone six to eight inches in size. They are laid in loose, but after the sand has run over them from half to three-quarters of an hour they will be packed quite tight. They are difficult to lay and to clean up, but are much more permanent than the wooden pavements. A patrol with a shot-gun loaded with bird shot, and a plentiful supply of salt and pepper in the charge, is a much better protection against theft. No one who has ever heard of the pain caused by the reception of such a charge would ever attempt to clean up for his neighbor.

The gold, quicksilver, and amalgam are caught in the spaces between the riffles. All the mercury is introduced into the top of the box and passes down with the rest of the material. The water should be at least two inches deep over the bottom. Boulders and stones must not be allowed to remain in the sluices, as they not only grind up the mercury and amalgam into flour, but are in danger of becoming wedged and obstructing the free flow of the water. They are taken out along the sluice whenever they are observed by the man whose duty it is to watch for them.

Sometimes a sluice is made without any boards by making a channel in the earth into which a stream is run. This channel is either in the pay dirt itself or in barren ground, the uneven bottom of which makes catch places for the gold. When a clean-up is to be made the water is turned off, and what is collected on the bottom panned. Only the coarse gold is saved; the fine is entirely lost. As it uses more water for the same amount of ground treated, it is only used in temporary workings. Such sluices are called "ground sluices."

When the rich dirt is below the level of drainage it is washed in a pit. There are two ways of doing this, depending for their application on the lay of the ground. One is to pump the water used in washing out of the pit with a Chinese pump, and the other to make a covered tail race through which the water can flow to a lower level, and cover it as the work progresses with the washed ground. This generally necessitates the use of a short sluice, or even only of a "tom." An active man can throw from three to seven cubic yards of dirt a day into the sluice, provided he does not have to lift it more than four and a half to five feet; at a less height he can do a little more. The quantity depends also somewhat on the character of the dirt which has to be moved. Sand without clay or boulders is most easily washed. Six cubic yards daily per man is fair average work. In a ground sluice where there is no lifting, but only shoving to be done, the average work per man will be about twenty-five cubic yards per day. When the sluice is worn out it is burned and the ashes treated for gold. If it has been used for some time the amount of gold recovered in this way will sometimes be sufficient to purchase a new sluice.

To make a clean-up the dirt is no longer put in at the head of the sluice. When the water runs entirely clear, five or six sets of riffles are taken up at the head of the sluice, their contents washed down to the riffles below. The sand is washed over, while the gold and amalgam lodge at the riffles and are removed. The next sets are then removed, and so on; sometimes twenty or thirty sets of riffles are taken out at a time. The work of a clean-up will generally take half a day to a day. It should not be made frequently, as it consumes so much time. It is generally done on Sunday. The length of the run will depend on the richness and character of the dirt. The amalgam collected is strained through a cloth and retorted.

When the dirt is rich and water cannot be had, the process of "winnowing" is used. For this purpose all the stones are picked out and the dirt is made as fine as possible. It is then tossed by two men on a hide or a blanket, with the object of having the fine particles carried off by the wind. After this has concentrated the pay dirt somewhat, it is again tossed on a "*batea*." Where the pay is below the surface, small shafts are sunk and galleries run short distances in all directions from it. The material brought to the surface is "winnowed." This is called

"coyoting," from its resemblance to the burrows of the coyote. None but the richest pay can bear this kind of work. Winnowing machines are now being made for use. The loss is enormous, but where they are used what is obtained is all gain, for without this method nothing would be had.

In very many of the deposits, after the loose sand has been worked, there remains below it a hard mass which was recognized as containing gold, but which could not be extracted. At first this was sent to the stamp-mill, but as all of it did not contain gold, and all the coarse boulders which were sterile had to be crushed so as to pass through the screens, as well as the fine material which contained the precious metal, this method was abandoned. Most of this hard material, which is called "cement," disintegrates when it is exposed to the air for a certain length of time, or is broken up by the frost. The cementing material is usually iron oxide, or clay. It was then proposed to work it by the hydraulic method, interposing in the course of the sluices a large number of "drops" and "grizzlies" and "undercurrents," in order first to break the cement by concussion, then to separate the larger sized boulders and catch the gold in the finer material.

When the deposits are very large such a method may be possible; generally, as it costs too much in time and labor, it will not be profitable. Disintegration by exposure and then washing in large revolving iron pans seems to be the most profitable method that has as yet been used. The pan bottom is perforated, and is so arranged that after a certain amount of the cement has been treated and all the fines extracted by water and passed into the sluice, a door in the bottom of the pan may be opened to let out the material too large to pass through the perforations into the sluices, so that it may be thrown out into the tails, which collect on the ground below. In the pan four arms with plowshare adjustments revolve to break up the cement, and make it fine enough to be carried through the bottom into the sluices. When the cement has been weathered or frozen and thawed, this work can be rapidly done. A pan 5 feet in diameter and 2 feet deep can treat from 40 to 120 tons of such cement per day, depending on the hardness of the gravel and the power available to move the pan.

Beach Mining.

It has been known from the very earliest times that gold occurred in river sands, and it is there that it is chiefly looked for in the early exploring expeditions in any country. That it existed to any extent in the sands of the sea beaches does not appear to have been known, or that sea water even contained a very minute portion dissolved in it, was not discovered until within a few years. The discovery of gold on the sea beaches of California and Oregon seems to have been like most of the other gold discoveries, the result of an accident.* A party of miners prospecting in northern California in 1850, of whom Eugene de Bertram was one, were forced by the jungle of the main-land to follow the line of the beach. In passing along the shore at what is now called Gold Bluff, in Klamath County, they found it dazzling yellow in the sunlight and covered with gold, which the natural action of the waves had separated from the beach sands, and so concentrated it that it yielded nearly fifty per cent. of precious metal. Collecting some of the sand they went first to Trinidad, about thirty miles distant, and from there to San Francisco, where a company was formed to mine the beach and collect the gold. Excited by the richness of the sample, and the truthful though somewhat exaggerated story of the miners, an expedition to Gold Bluff was formed, and started from San Francisco in a revenue cutter. In order to keep the secret from the people of Trinidad, they anchored off Gold Bluff, and endeavored to land on the spot where the gold had been found. The first boat contained six persons, but on nearing the shore it was swamped and all were drowned but de Bertram, who was rescued by the adroitness of a friendly Indian, who happened to be on the shore at the time. The rest of the party returned to Trinidad, and went on foot to the beach. But there was no gold there, and no trace of it could be found. Discouraged and disheartened, most of the party went back to San Francisco, but de Bertram, though weary and almost hopeless, yet believing that what he had once seen he could find again, waited and watched until a favorable wind and tide washed the gold up again. The result of this discovery was that soon the beaches were explored, and have since that time been worked with moderate profit to the owners; not sufficient, however to cause any very great excitement or the investment of any

* Min. Com. Report, 1870, p. 87.

very large amount of capital. As the concentration is done without cost, and the troublesome question of tails does not have to be taken into consideration, since the sea takes care of them, it promised to last into the distant future, and to give a rich return for the labor and capital expended. The excitement caused by their first discovery soon received a damper, for with the rude and careless methods by which they were worked, not more than one-fifth of the gold was separated, and the uncertainty of the winds and the waves bringing any up for a considerable length of time, for it was necessary that the wind should come from a certain direction, to cause the light particles of gold to be brought up on the beach, soon proved that no great fortunes were to be made here. These beaches were apparently soon exhausted and for a number of years were abandoned, on the supposition that they were no longer worth working. Many persons, supposing that these golden sands came entirely from the deep sea, undertook to dredge and to work the sands with complicated and expensive machinery, which, however, soon proved to be an entire failure.

These beach-sands occur in the northern counties of California and the southern counties of Oregon, extending from Cape Mendocino to the Straits of Fuca. They are also found in small quantities as far south as San Francisco, and as far north as Puget Sound, but in too small quantities to pay for working, the district where it pays to work being confined to the above limits. The diggings extend along the coast for three or four hundred miles. The bluffs vary from 110 to 500 feet in height. They will average 200 feet. Most of them are vertical, but they are sometimes overhanging and have very narrow beaches at their foot. High up on the beach, where the action of the waves is very gentle, the sand is found in alternating layers of the fine material composing it, with the lighter material on the top. When, on a windy day, the light sand of the upper surface becomes dry, it is often blown away to such an extent as to leave the heavy black sand on the surface. Lower down, however, where the surf breaks, the sand is coarse, and in heavy weather may be composed almost exclusively of pebbles. In California, as in New Zealand and Australia, these gold sands sometimes extend considerable distances into the ocean. Opposite Crescent City, ten miles out to sea, gold was found in the soundings brought up by the lead, but all attempts to dredge it have failed. These sands have doubtless been deposited by some

ancient river flowing through the gold region of Oregon and California, which for ages carried in suspension, in its current, immense quantities of auriferous gravel, and which gradually filled its bed and the surrounding country with its deposits and subsequently became elevated to its present position above the surrounding country.

In some cases the whole length of the bluff is gold-bearing, with rich and poor streaks, yielding on the average 20 cents to the cubic yard, but the rich streaks giving as much as $135 to the ton, of gold .957 fine.* The sand contains gold and platinum in small quantities, in minute scales, most of which are flat, mixed with a very large percentage of black, magnetic, and ordinary quartz beach-sand. The shape of the grains, which can be seen when they are highly magnified, is oblong and doubly convex, with edges thicker than the middle, a form which would naturally result from the continued pounding of the grains. Their thickness is rarely more than 0.001 of a line. In length they vary from 0.3 to 0.7 line. Much of the black sand is composed of grains of nearly uniform size, but they are rounded, so that although the gravity of the sand is only one-fourth that of gold, the buoyancy of the latter counteracts this, and it is possible to separate them. The gold is not only buoyed up by the air, but is as easily moved by the water as the black sand itself, so that it requires the greatest skill to obtain "color" either with the shovel or the pan.

It was at first supposed that these beaches were confined exclusively to the coast-line, but it has since been discovered that older beaches containing the black sand exist as much as thirty miles back of the coast-line, and often at considerable elevations, and in such a position as to show that their elevation must have been very gradual. The black sand containing the gold is sometimes covered to a depth of from three to sixty feet with sterile material. In Oregon these are called "back beaches;" they are worked either by stripping or tunneling.

All the early methods of extracting gold were based on the difference in specific gravity. The best of them did not save more than 20 per cent. of the gold. Plattner's process was tried, but was found to be too expensive to be used on a large scale, but the use of it for a short time confirmed the fact that fully 80 per cent.

* Prod. Gold and Silver in the U. S.: Burchard, 1884, p. 557.

of all the gold contained in the sands was being lost. When these sands contain gold that can be seen by the microscope, most of it is quite bright, though it is covered occasionally by a coating which prevents the contact of the mercury. In most cases this appears to be of an organic nature, and seems to have been produced subsequently to the formation of the bluff.

The remarkable purity of the gold obtained from the beaches has attracted a great deal of attention. This is undoubtedly owing to the fact that this gold being in an extremely finely divided state, and wet with the salt water for so long a time, has had the silver separated from it in the state of chloride, as is also the case with the other metals, leaving the gold almost pure. The value of these sands is very variable, depending on the situation and the state of the natural concentration at any particular time. Values of from $10 to $30 a ton are quite common, but the sand often has less, and sometimes contains many times this amount.

As the operation of mining is entirely dependent for success on the state of the weather, the direction of the wind, and the force of the tide, it is always uncertain, but as it requires very little outlay for plant, and not a great amount of labor, it is quite attractive; but as the concentration of the gold in sufficient quantities to work is dependent on the weather, the best claims are often unworkable for months, during which time the miner must wait patiently for any change of wind which will throw the sands up on the beach where he can reach them. He may not even be rewarded for many weeks for his patience in waiting by any days of favorable winds, or even by a single tide. The sea-water cannot be used without great loss for concentrating by hand, as it is too heavy, so that the beach miner must wait for nature to do the concentrating, which, when once effected and collected for treatment, may be entirely washed away in a very few minutes by a contrary wind or a heavy tide. Such uncertainties do not invite the investment of any large amount of capital.

A favorable wind, with the heavy swells at high tide, drives the sand upon the beach. The ebbing tide, as it recedes, washes out the fine particles of rock, the lighter materials retreating with the water, leaving the heavier ones, the iron, the gold, and the platinum, high up on the beach. When the tide is out, this sand can be scraped up and "packed" inland out of the reach of the waves, to be treated there. In the intervals between two tides, or when

sufficient has accumulated, the sands are washed in a sluice, terminated by a tom, with the ends and sides turned up. The bottom is perforated with holes, and underneath is the slightly inclined trough with silver-plated amalgamated copper plates. The sand, passing through the holes, strikes the plates; the heavy particles of gold are caught on the amalgamated plate and by the mercury, while the sands and the float-gold run off at the end of the sluice. The method is imperfect, all the more so as the iron-sand is more difficult to remove from the buoyant particles of the precious metal which characterize these sands, and which are very much lighter than those found in ordinary placer mining, chiefly on account of its state of very fine division. Much of it floats, and can be made to sink only by standing for a very long time, or by heating the sand, which is entirely impracticable in a large way. The probable quantity of gold is very uncertain, for a single storm will often wash out all the sand to the depth of 6 or 8 feet, leaving the rocks entirely bare.

It has been supposed by some that this gold is cast up from the depths of the sea, but this theory does not seem to be a probable one, for the reasons that the attempts to dredge it in deep water have not been successful, because all the bluffs which skirt these beaches contain gold, and because the black sand which covers the gold is also found in a number of places several miles inland, at a height very much above the present level of the sea. There are indications that the rivers have in former times been blocked by landslides, which have sufficed in some instances to turn them from their courses into other channels, and that the rivers themselves have carried down immense deposits into the sea, where the black sand, the gold, and the platinum in the more tranquil waters would tend to separate by gravity. There is also abundant evidence that the front of all the cliffs is gradually receding, and as they contain gold in sufficient quantity to be worth working, this sufficiently accounts for the presence of gold in these beaches. It is also constantly observed that after a very heavy storm which undermines the bluff, and causes large quantities of fresh material to fall, the sands are not only richer but the gold is coarser than before. Companies have several times been formed to work the bluffs, where they were rich enough to be worked. This would seem to be a very attractive enterprise, since the bank could be cheaply blasted and the sand hydraulicked, while the

sea would do all the concentration and would at the same time dispose of all the tails. It has found many enthusiastic adherents, but has, however, never been extensively tried, for the distance from which fresh water would have to be brought to wash the sand, sometimes 40 or 50 miles, makes the process impracticable. In many cases, especially in the interior of the country, the gold of the black sand, which is very fine, does not readily amalgamate until after it has been heated, which seems to destroy a film of organic matter with which it is coated, and which has consequently prevented the contact of the particles of gold with the mercury. It has been proposed to treat these sands with Plattner's process, but this has also been found to be generally impracticable, although the attempt to do so revealed the fact of the enormous losses that were being made by treating them by any other process.

It was very generally supposed that in the first mining the greater portion of the gold contained in the beach-sand was saved in the appliances that were used to collect them, and subsequently that the beach-sands themselves had become exhausted. But careful assays have since proved that by far the larger part of the gold was lost in all the earlier workings in the state of an impalpable powder, and in such small particles that it is scarcely visible with the microscope. How to separate these fine particles of gold from the heavy black sand has been a problem which if the sands had been richer would undoubtedly have been solved. The extreme lightness of the microscopic particles of gold, and the much greater weight and larger size of the grains of sand make it impossible to separate them by water or air, and in some cases the gold itself is coated with some substance which prevents the mercury attacking it. As there are millions of tons of sand that contain from five to ten dollars in gold to the ton, and are sometimes as high as $100, the experiments which were made to save it by chlorination and other chemical methods, confirmed the losses which were being made in the working of the sands, but suggested no practicable method of treatment.

One of the earliest worked and most famous of these localities is at Gold Bluffs in North California. The coast-line after leaving Klamath River is very broken and rocky. The sands appear at Gold Bluffs about four miles south of the river, where for a long distance there is almost an unbroken line of cliffs varying from 100

to 500 feet in height. Some of these bluffs are vertical, and, in some cases, overhanging. They are composed of regular horizontal strata of beach-sand, gravel, and boulders. Beneath these are the trunks of very large redwood trees, lying horizontally, and stumps standing in their natural position, but below the present ocean level, showing a change of level both in the land and sea. At low water there is a narrow beach, but when the tide is full the sea washes the base of the cliffs. The method of working is to wait until the surf strikes the beach at an angle.* When the direction of the wind is such that the surf breaks perpendicularly on the beach, masses of coarse gravel are thrown up, and no black sand, but when the waves strike the beach at an angle, the gravel is washed into heaps in certain places, and in others black sand, more or less rich in gold, is deposited. During heavy weather the beaches cannot be worked, as they are usually too narrow, and the action of the waves prevents the deposition of the black sand. During a severe storm, the whole surface is covered, so that the surf strikes the base of the cliff, and when it subsides the shape of the beach will usually be entirely changed. Generally, the sand containing the gold lies on the top, but sometimes the black sand is covered over with a deposit of barren material, which has first to be scraped away before the black sands can be collected.

At Ophir Beach, near Rogue River, Oregon, the beach below high water is washed in a small way by the sluicing process, when the sand that is thrown up is rich enough to pay. The superintendent examines the beaches every day, collecting the sands and scraping up the loose gravel; he pans it, and on finding a "prospect," sends for a mule train, each mule carrying two coarse hide or canvas bags, which will contain about 125 pounds each; a train of 40 mules would, therefore, transport 10,000 pounds, or five tons at a single trip. In order to collect the sand the top gravel is stripped and the black sand gathered into piles. The canvas sacks are taken from the mules and filled, the mules being trained to start on a run for the works the moment they are loaded. As in some places the sea comes close up to the cliff, the animals are trained to turn their heads toward the cliff, bracing their feet firmly, allowing the wave to dash over them, and then going on their way on a run till the next wave comes. During a single tide, these

* Mining Com. Rept., 1874, p. 145.

mules, under favorable conditions, are able to make three trips, when the distance to the working place is not too great. At the works the sand is piled up and washed in rockers, long toms, or sluices, with amalgamated plates.

At Randolph, in Oregon, at the mouth of Whiskey Run, the gold in the sands was always very fine. No nuggets of more than four to five cents in value were found, but the sand was very rich. Only a small portion of the gold was saved. Much of the ground was worked over six or seven times in succession, and was each time found to be remunerative. For a very short time a single miner could earn a thousand dollars a day,* but the prosperity was of short duration. The town has been in ruins for many years. The largest amount ever taken out of any of the California beach-claims is said to have been $25,000 in one year. The great difficulty in working the beaches has always been the lack of fresh water. They have been washed with salt water pumped from the sea, but the amount of gold collected in such cases was only a small portion of what was lost.

All the experience of these beach mines goes to show that immediately after a heavy cave or slide on the banks, the beaches are rich in coarse gold, but up to the present time it has not seemed profitable to do any bank-blasting or to undertake mining on a large scale by the hydraulic processes. The gold is washed up by the sea only. Many suggestions have been made that the gold would be found in the sea beyond the surf-line, and some attempts to dredge for it have been made, but unsuccessfully.

It was proposed at one time to effect the amalgamation of these extremely light particles before endeavoring to concentrate them, by heating the sand and quicksilver under water up to the boiling point, which would cause the mercury to diffuse itself through the entire mass, and thus attack the gold that was in a proper condition to amalgamate. If this sand had been previously calcined, the first calcination and the subsequent heating would be sufficient in most cases to remove the coating upon the gold. In some cases a chemical treatment of the sand was made, and after all this was done, the fine particles of amalgam had still to be collected by boiling, and unless they were very rich and fuel plenty, it is evident that such a method as this, although patented and supposed to be

* Min. Com. Rept., 1873, p. 207.

worth a great deal of money, could not be successful. The process* consisted in screening the sand and subjecting it for twenty-four hours to a solution of caustic potash and salt, for removing any coating upon the gold, and destroying the compounds of sulphur which would attack the mercury. The material so treated is heated in a pan by a jet of steam, and the quicksilver poured in; it was kept constantly in agitation by the steam, and the movement kept up from a quarter to half an hour, by which time the gold should have become thoroughly amalgamated. The pulp was discharged into a vat to cool. It is desirable that it should not be heated too much, as that divides the quicksilver more than is necessary to secure the amalgamation. Care must be taken to use pure quicksilver in the process. The cooled material was then passed over a series of copper rollers grooved spirally and set, the one above the other. These are set over a sluice, three feet wide, with amalgamated copper plates and grooved copper rolls. The rollers are hollow, 1 foot in length, and 2 inches in diameter. As the amalgam accumulates on the rollers, it is either cleaned off or drops into the sluice below. Notwithstanding all that was hoped for it, the process was not successful.

It does not seem likely that beach mining will ever be very remunerative; the uncertainties connected with the value of the sands, and the times when the beaches can be worked, for they will often be unworkable for months together, either because the storms are so high as to denude the beach, or the weather so calm that there is not force enough in the waves to throw up the black sand on the beach or wash it down from the cliff. It is not probable that it will ever justify the investment of a large capital. That it will, however, yield a fair return for a small outlay, is generally true. Persons residing in the vicinity of these beaches have often made a fair living by working them when they could be worked, but no company, whatever its prospect at the commencement, and however rich the beach when they began to work it, has ever been able to work these sands with profit for any considerable length of time. There are places where if fresh water in sufficient quantities could be had, the bluffs might be worked by hydraulic methods, but even this would not be justified if the line of ditch was to be very long, or if the storage of the water required the con-

* Min. Com. Rept., 1875, p. 317.

struction of large dams. The immediate disposal of the tailings by the sea looks at first sight very attractive, but it remains to be seen whether the accumulation of enormous amounts of débris would not stop the encroachments of the sea, and whether in working such fine gold on an enormous scale, sufficient of it could be caught to make the enterprise profitable.

River and Bar Mining.

Bars are the deposits that form in streams which bring down auriferous gravel, when the stream strikes a hard rock, and is deflected to the other side of its course. Parallel to the line of deflection, at an angle more or less great to the course of the stream, bars will form, on and in which the gold lodges. In a modern stream, these bars will, sometimes, rise above the surface, and are worked in a small way. The ancient rivers, however, formed bars upon a much larger scale. Such bars, running across the channels of these ancient streams, most of which have disappeared ages ago, can be worked with great profit by hydraulic methods. They are found where the river still retains its ancient course, but the diminished size of the modern stream will frequently make these bars from 40 to 50 feet above the present high-water level. They are ordinarily worked by the hydraulic methods. Where the bars are simply in the rivers of the present time they are of local and transient interest only, as they are very soon worked out. The gold they contain has sometimes been so concentrated by the action of the stream that a single panful has been known to yield more than $100. They have been pumped and dredged, but they do not generally justify any large expenditure, and all workings based on any large yield for a period of time have been unsuccessful. Such plans were proposed at Oroville on Feather River, California, but the failure was as great and disastrous as the expectations were large and promising. The Chinese are, however, working these deposits and make a fair living out of these abandoned bars or their tails.

In very many rivers in California and elsewhere which run through alluvial deposits containing gold, or which have abraded the outcrops of veins containing it, there is a deposit more or less large of gold in their beds. In this case, which is very common in California, two methods are used. One is to dredge, or pump the sand from the river bed or to undermine it, and another is to

remove the whole of the water from the river by means of a flume and "flutter wheels," and then to work the ground of the river bed and beneath it. Such experiments as these are extremely hazardous, as the least defect in the construction of a flume or a freshet may break up the whole of the works, the outlay for which must of necessity be very large. They can, therefore, only be undertaken during the summer, when the water is low and when there is no danger of sudden rises in the river. Flumes commenced before the spring rains were over have more than once been carried away before they were finished. Usually these river claims, as they are called, are kept dry from leaks in the dams and from the water of small streams which enter the sides by wheels which work in the flumes constructed to carry off the water by the force of the current.

On Feather River,* in Butte County, California, on the "Cape claims," two wing dams were constructed across the river. All the water above the first dam was carried by a large flume to the lower one. The water between the two dams was pumped out by wheels run by the current in the flume. The gravel on the river bed was washed in sluices, and during forty-two days yielded $680,000. Only one-third of the ground had then been washed out when the flume was destroyed. Such methods can only be employed on small streams in the dry season when there is the least water to contend with. The stream must be diverted from its course or pumped out. Such methods, as shown by the example cited above, are full of risk. They seem very attractive when the rich pay dirt is brought up from the bottom and the calculation is made of how much gold is contained in the river bed, but it is seldom worth while to flume them for a short distance,—generally it will not pay to do it for less than three or four hundred feet, and often more must be flumed,—and when all the water has been pumped out, derricks run by hurdy gurdy wheels or by hand power must be erected to remove the large rocks to get at the gold beneath them, with no certainty that the whole of the work will not be swept away by a sudden rise in the river.

When such a method is not practicable, the river is explored to find the bars and rich placers. When they are located, a shaft is sunk to the depth of 40 to 60 feet above high water-level, and a

* Prod. Gold and Silver in U. S., 1880, p. 328.

drift run in the rock under the bars. The gravel is taken out from above, hoisted to the surface, and washed in sluices. The water pumped from the shaft, can, if necessary, be stored for washing. Gravel containing only seventy cents a cubic yard is worked in this way.

Where the quantity of water is, or is likely to be, small, in order to economize it, it is the custom to dam the stream, and, when the water has accumulated sufficiently, to open the gates and allow it to rush out against the gravel with great force. This process, called "booming," is sometimes carried on by hand power and sometimes automatically. When the process of "booming" is carried on automatically, it is effected with the use of the so-called "shot-gun dam." This is an automatic discharge gate, so constructed as to have the gate of the dam lifted at intervals by the water itself; to do this the gate is attached to one end of a suspended beam, which has a water-tight box with a valve in the bottom, attached to the other. When the dam is full a part of the water flowing over passes into a launder, arranged for the purpose, which conducts the water to the box. This is so adjusted that when it is full it lifts the gate by itself sinking. The contents of the reservoir is thus instantly released, and flows with great force and rapidity to the point where it is to be used. After it has fallen a certain distance the valve of the water box comes in contact with an upright post, which opens it and lets the water out. The empty box rises and closes the gate. This is repeated at regular intervals, giving a heavy head of water and a rapid current every few minutes. This method has been extensively used both in Colorado and California. Before this method was invented the gate was always lifted by hand and required the constant attendance and care of one man. By the present method the gate requires to be looked after only occasionally. The work is entirely automatic, the number of times per hour that the gate is lifted being perfectly regular and depending only on the supply of the water.

It has been proposed, and a great many attempts have been made, to raise the gold contained in the river gravels from the bottom of the streams with powerful pumping engines, capable of raising 25,000 cubic meters in the course of twenty-four hours, half of the bulk being water. It is claimed that these pumps are cheaper and better than any dredging machines. Notwithstanding these assertions, however, neither dredging nor pumping has proved

successful, for the reason that the river can only separate the gold in the same way that a sluice does. The idea of most persons not familiar with river sands, is that the sand is deposited in regular stratified layers, and that the gold is disseminated regularly through them. No one who has any experience in sluice mining would believe this, for the whole object of the sluice is to provide places where the gold may be caught in mercury, the bottom of the sluice being arranged with reference to this. At the bottom of the river the sands are always unequally distributed. The bed rock is never even, but is always filled with crevices, into which the gold falls by gravity and is not accessible to the dredge. Where the river flows over a soft gravel, and the bed rock is either turned on its edge or is soft from decay, it is well known to gravel miners that the gold will penetrate the bed rock sometimes for a considerable distance, so that it is frequently worth while to take it up for some depth. Dredging and pumping machinery must do their work progressively and systematically, starting from a fixed point and proceeding toward some other. It is impossible for the machine to find where the gold lies, and even if every spot were known beforehand, the irregular nature of the deposit, and the fact that the bed rock is often the richest part of it, would make it impossible to work a spot here and there, so that in pumping or dredging the barren material will be more likely to be raised, even supposing the gold is actually there, than that which contains it, as the gold is hidden away in pockets beneath the ground, which can be reached by the dredge. These processes, therefore, have not been successful, and it may be stated as a general proposition that if the river bed is poor it will not pay to work it by any method, and if it is rich the usual process of damming and fluming, which has sometimes proved successful, with all its uncertainties, is probably the best, as it is certainly the cheapest method of working it in dry seasons when the flumes are properly constructed. In such a case the practiced eye of the miner distinguishes the points where the gold is likely to be found and works those points only, leaving the barren material behind, but the machine must take poor and rich as it comes, and cannot in any case work the bed rock.

It would be impossible to state any yield for the placer deposits as an average. Those which have been published were interesting at the time because they were extraordinarily large, while the poor yields have not been made known. The few successes have been

heralded all over the world, but the great army of unfortunate ones has not been heard from, and scarcely excites any commiseration or sympathy. To average the few successes of the fortunate with the immense number of failures would bring the gains down so very low that few thinking men, if they were to consider the proposition by itself, would care to run the risks of placer mining, but every one hopes that he will be one of the fortunate ones, and hence many make the attempt. It is doubtful whether, over a large and moderately rich district averaging successes and failures, the yield is as much as ten dollars a day for each man, and this, with the great cost of living in the places and times when such mining is possible, is scarcely more than a bare living. How many men never found enough to keep their hearts up, and died, either in the field or the insane asylum, from grief over the loss of all they had saved, or from disappointment at never having realized their dreams of wealth, we shall never know, but the number must have been very large to bring the average down so low. James Marshall's life and death are an excellent illustration of the proverb: "He that hasteth to be rich hath an evil eye, and considereth not that poverty shall come upon him."*

TURNOUTS FROM COMBINATION TRACK.

BY EDGAR G. TUTTLE.

WHERE railroads having different gauges make connections, they frequently use the same road-bed in common. The result is that we have a combination track or a combination of gauges which may be of three or four rails, as follows:

Fig. 1. This is the simplest form of the combination of two different gauges with three rails. One rail being common to both gauges.

Fig. 2. Two different gauges with four rails, having a common centre.

Fig. 3. Two gauges with four rails, not having a common centre.

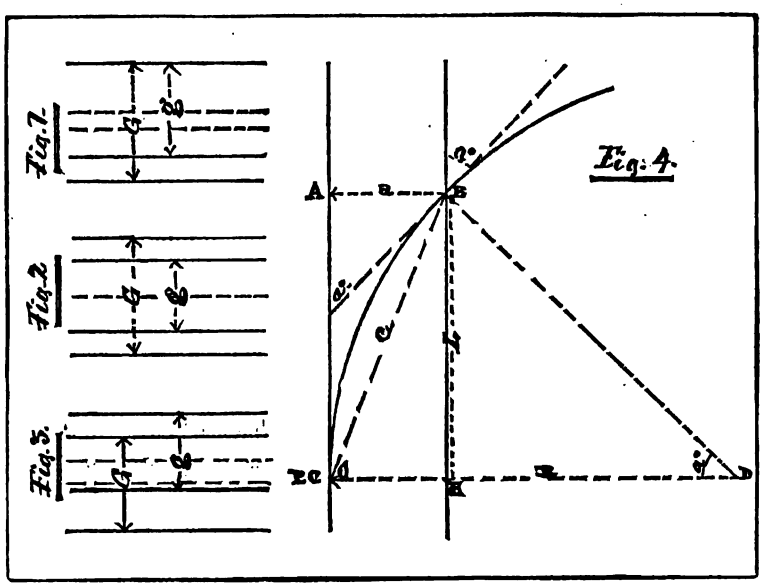
These three cases will be treated as they occur in, I. Turnouts

* Proverbs, xxviii., 22.

from straight track. II. Turnouts from curves in opposite direction. III. Turnouts from curves in same direction.

I. Turnouts from Straight Track.

To determine the leads, frog angles, etc., of a turnout in any of these cases, we can apply the formula for the tangential deflection (Henck, *Prob.* 17 and 18), as follows:



In Fig. 4 let R = C D = radius of curve C B and d = A B = tangential deflection for a given chord C B = c.

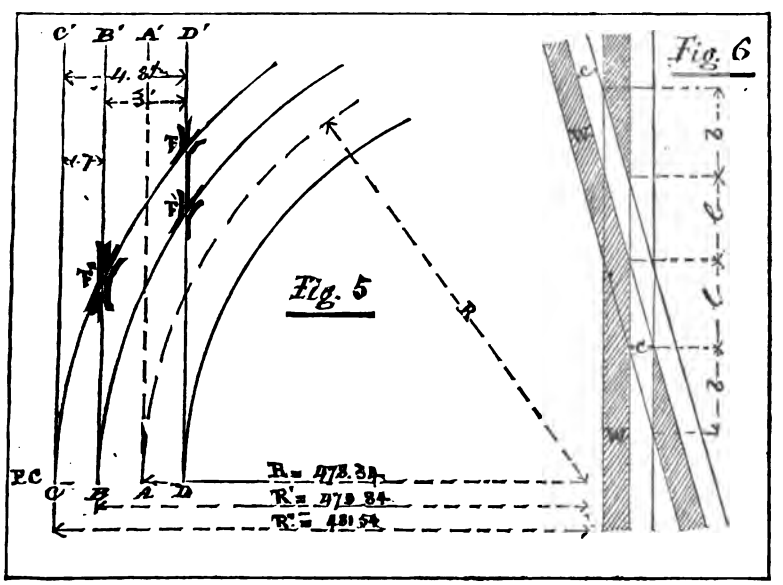
Then we have $d = \frac{c^2}{2R}$ and $c = \sqrt{2dR}$. This is a formula for finding the chord when the tangential deflection and radius are known. Let A C represent the tangent to the curve at C, and through B draw B E parallel to A C. Let A C and B E represent straight track, and C B the curve described by the outer rail of the turnout with a radius R, crossing the rail B E at B, which is the point where the tangential deflection becomes equal to the gauge. Then we simply have to find the length of the chord c according to the different values of d (which represents any gauge). In Fig. 4 we see $c = \sqrt{L^2 + d^2}$, combining this with $c = \sqrt{2dR}$ we have $L = \sqrt{(2R - d)d}$, making d = g (the gauge) in the above we

have the following formula for the lead as measured from the point of curve (P. C.) to the point of frog (P. F.).

$$L = \sqrt{(2R - g)g}, \quad \ldots \ldots \ldots \ldots \ldots \quad (1).$$

This is a formula of general application to all cases of turnouts from a straight track. In laying out switches the frog angle is generally the given quantity and the radius is calculated. In the case of combination track the radius is generally given and the frog angles or numbers deduced. In a curve with a large radius if the calculated frog numbers come within one-tenth or two-tenths of the standard numbers of frogs, these may be used by spraining them until they conform with the numbers required in the turnout, and spiking fast to the ties in this position.

First Case of Combination Track: Example.—Required the leads, frog angles, etc., in a combination turnout with three rails. Gauges = 4′ 8½″ and 3′. Turnout curve 12° to right. See Fig. 5.



Let AA′ represent the centre line of the narrow gauge (3′) track BB′ DD′, and CC′ DD′ represent the standard gauge (4.7) track, DD′ being common to both gauges. The above lines represent

the gauge side of the rails, and the measurements given are from gauge to gauge.

R = radius of centre line of turnout = 478.34 feet.
R' = 478.34 + 1.5 = 479.84 feet.
R'' = 478.34 + 1.5 + 1.7 = 481.54 feet.

The values of g and R in formula (1) depend upon the lead of the frog we are to determine. Thus, for any frog, R is equal to the radius of the turnout rail (= radius of centre line of turnout ± distance of turnout rail from centre line), and g is equal to the perpendicular distance from the turnout rail under consideration, at its point of curve (P. C.) to the rail it is to cross. As in the case of the lead for the point of frog F, g = 4.7; for F′, g = 3′; for F″, g = 1.7′. Substituting the above values of R and g in formula (1) $L = \sqrt{(2R-g)g}$, we have

Lead DF $= \sqrt{(2\times 481.54 - 4.7)4.7} = 67.11$ ft.
" DF′ $= \sqrt{(2\times 479.84 - 3)3} = 53.57$ ft.
" BF″ $= \sqrt{(2\times 481.54 - 1.7)1.7} = 40.42$ ft.

The lead BF″ is the distance from the point of curve of the turnout to the throat of the double-pointed frog. In Fig. 4 we see the frog angle is equal to the angle at the centre of the circle D, whose radius is R, and subtended by the chord c. Hence we have the following formula for the frog angle (a):

$$\text{Sin } a = \frac{L}{R} \quad \ldots\ldots\ldots\ldots\ldots\ldots \quad (2).$$

Substituting in this formula the values of L and R from above, we have for

frog F, $\sin a = \frac{67.11}{481.54} = .1394 \therefore a = 8°1'$.
" F′, $\sin a' = \frac{53.57}{479.84} = .1116 \therefore a' = 6°24'$.
" F″, $\sin a'' = \frac{40.42}{481.54} = .0839 \therefore a'' = 4°49'$.

The frog number (n) is determined by the number of times the width of the heel is contained in the length of the frog. If we denote the width of the heel by unity, and the length of the frog

in terms of this unit by n, and the frog angle by a, then tan $\frac{1}{2}$ a $= \frac{\frac{1}{2}}{n}$; transposing we have the following formula for the frog number:

$$n = \frac{1}{2\tan \frac{1}{2}a} \quad \ldots\ldots\ldots\ldots\ldots\ldots \quad (3).$$

Substituting the frog angles found above ($a = 8° 1'$, $a' = 6° 24'$, $a'' = 4° 49'$) in formula (3), we find for

$$\text{frog F } n = \frac{1}{2 \times .07007} = 7.13$$

$$\text{“ F' } n' = \frac{1}{2 \times .05591} = 8.94$$

$$\text{“ F'' } n'' = \frac{1}{2 \times .04205} = 11.9.$$

In the above cases we can use the standard makes of frogs, viz., numbers 7, 9, and 12. Approximate formulæ for finding the frog numbers are

$$n = \frac{L}{2G}, \text{ and } n = \sqrt{\frac{R}{2g}}.$$

In the same way that formula (1) for the lead is deduced, a formula can be obtained for the length of the switch rail. It is simply necessary to find at what point along the curve C B (Fig. 4) the tangential deflection from the line A C becomes equal to the throw.

In the formula for the tangential deflection, $d = \frac{c^2}{2R}$, making d = t, the throw, and c = l, the length of switch, we have the formula for the length of switch rail

$$l = \sqrt{2tR} \quad \ldots\ldots\ldots\ldots\ldots\ldots \quad (4).$$

In the above formula l is measured along the chord. The difference in the length of l is not appreciable, whether it be measured along the chord or parallel to A C. (Fig. 4.)

To obtain the theoretical length of the switch rail (denoted by k) as measured parallel to A C we have $l^2 = k^2 + t^2$. Combining this with (4) we have $k^2 + t^2 = 2\,t\,R$ and

$$k = \sqrt{(2R - t)t}.$$

Suppose R = 478.34 ft. and the throw t = 5 in. = .42 ft. Substituting these in (4) we find l = 20 ft. This is the length of the switch rail or distance from point of curve of turnout (P. C.) to head block (H. B.) in the case of a stub switch. If the switch rail does not conform with the turnout curve when thrown, it may be necessary to spike it for about one-quarter of its length from the P. C. towards the H. B. Or if the whole length is free, it may be forced to curve as desired by driving spikes in the ties, against which the rail bends when thrown in position. In the case of spring or flat switches, the throw being at the point of curve, the rails would be spiked or forced to a curve, as above stated, but the spiking is done from the frog towards the P. C. as far as is required.

The construction of the double-pointed frog F″ (Fig. 5) may be determined either graphically or by calculation. The figures obtained by calculation are more accurate. In this case the crossing angle is 4° 49′; corresponding frog number = 11.9. In Fig. 6

Let w = width of the rail;
l = distance from throat to either point of frog;
c = clearance;
b = length of bevel for pointing a rail;
n = frog number.
then $l : c :: n : 1 \therefore l = c \times n$ . . . . . (5),
and $b : w :: n : 1 \therefore b = w \times n$ . . . . . (6).

Suppose c = 2 inches and w = 2⅛ inches, then l = 23.8 inches and b = 25.3 inches.

Second Case of Combination Track.—Here where there are two gauges with four rails having common centre, we proceed in the same way to find the leads, frog angles, etc., as in the first case.

It is often necessary to curve the frogs when they are very long or where the turnout is very sharp, the heel of the frog being thus made wider and the throat ends being narrowed. By application of formula (1) and a table of ordinates the curvature can be found.

Fig. 7 shows a turnout from a straight line in the second case of combination track. The lines represent the gauge side of the rails, and the measurements are from gauge to gauge.

Example.—Required the leads, frog angles, etc., in a combination turnout with four rails, having common centres. Gauges = 3′ and 20″. Turnout curve 12° to right.

In Fig. 7 the different values of g and R are given, the radius of the centre line of the turnout being 478.34 feet (12° curve). Recollecting that g is equal to the perpendicular distance from the

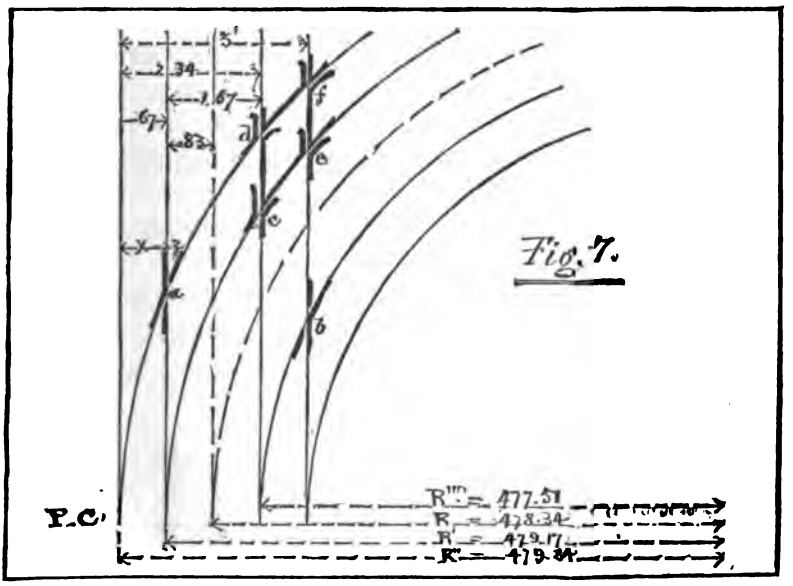


Fig. 7.

turnout rail under consideration at its point of curve to the rail it is to cross, and substituting the values of g and R from Fig. 7 in formula (1), we have the following leads:

$$\text{For frog a, } L = \sqrt{(2 \times 479.84 - .67).67}$$
$$\text{" " b, } L' = \sqrt{(2 \times 477.51 - .67).67}$$
$$\text{" " c, } L'' = \sqrt{(2 \times 479.17 - 1.67).167}$$
$$\text{" " d, } L''' = \sqrt{(2 \times 479.84 - 2.34)2.34}$$
$$\text{" " e, } L'''' = \sqrt{(2 \times 479.17 - 2.34)2.34}$$
$$\text{" " f, } L''''' = \sqrt{(2 \times 479.84 - 3)3}$$

Substituting the above values of L and R in formula (2) we have the following for the sines of the frog angles:

$$\text{For frog a, } \sin a = \frac{L}{479.84}$$
$$\text{" " b, } \sin a' = \frac{L'}{477.51}$$

$$\text{For frog c, } \sin a'' = \frac{L''}{479{\cdot}17}$$

$$\text{“ “ d, } \sin a''' = \frac{L'''}{479{\cdot}84}$$

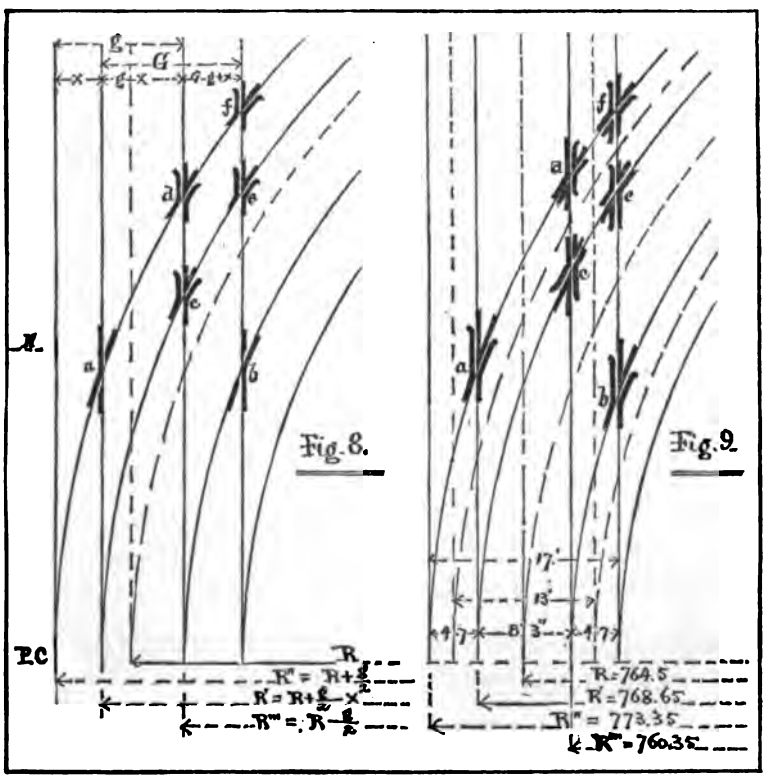
$$\text{“ “ e, } \sin a'''' = \frac{L''''}{479{\cdot}17}$$

$$\text{“ “ f, } \sin a''''' = \frac{L'''''}{479{\cdot}84}.$$

Similarly we can find the frog numbers by substituting the values found above in formula (3).

The length of the switch rail can be found, as already explained, by the use of formula (4).

Third Case of Combination Track.—Here we have two gauges with four rails not having common centres. These gauges may be



equal or unequal (see Fig. 8). R is the radius of the centre line of the track with a gauge g; the other gauge is denoted by G. The

distance between the gauge sides of the two left-hand rails is denoted by x, the other relations between the gauge side of the rails are then found to be g—x and G—g + x, as shown in Fig. 8. The values of R′ R″ R‴ can then be found. Proceeding in the same manner as in the former cases, substituting the above values of R and g in formula (1), we obtain the different values of L. Then substituting the different values of L and R in (2) we obtain the frog angles, and so on applying formulæ (3) and (4).

To find the leads, frog angles, etc., in a double track turnout from a double track, we can apply formulæ (1), (2), etc. Suppose the radius of the centre line of a double track R = 764.5 and the gauge G = 4.7 and the distance between the centres of tracks d = 13 feet (see Fig. 9, where the different values of R and g are shown). Substituting these in formula (1) we have the following leads:

$$\begin{aligned}
\text{For frog a, } L &= \sqrt{(2\times 773.35 - 4.7)4.7} \\
\text{“ “ b, } L' &= \sqrt{(2\times 760.35 - 4.7)4.7} \\
\text{“ “ c, } L'' &= \sqrt{(2\times 768.65 - 8.3)8.3} \\
\text{“ “ d, } L''' &= \sqrt{(2\times 773.35 - 13)13} \\
\text{“ “ e, } L'''' &= \sqrt{(2\times 768.65 - 13)13} \\
\text{“ “ f, } L''''' &= \sqrt{(2\times 773.35 - 17.7)17.7}
\end{aligned}$$

Then substituting the above values of L and R in (2), proceed as in former cases for the frog angles. Then apply (3) for the frog numbers and (4) for the length of the switch rail.

As may be seen from the above application of formula (1) etc., the values that can be assigned to g may be so great as to cover all cases that arise in the calculation of the leads, frog angles, etc., in any number of parallel straight tracks with any number of turnouts of whatever degree—the curves not intersecting, of course. In the case of combination track the double pointed frogs (crossing frogs) can be dispensed with by introducing the throw at H (Fig. 8), and similarly in Figs. 5 and 7, provided the throw required is not too large. Throws as large as 15 inches are in use, and, perhaps, greater, but on account of the friction of the switch-rail and also its length and other causes, a small throw is sometimes desired.

To find the length of a switch-rail with a large throw, so as to do away with frogs a and b (Fig. 7), proceed as follows: Let the dis-

tance between the two left-hand rails be denoted by x (x = .67). Then z, which is equal to about 5 inches in the case of standard-gauge rails, or 4 inches in the case of narrow-gauge rails, represents practically the nearest that the gauge side of 2 rails can approach each other on the head-chair. So that x + z = throw required.

Fig. 7 represents narrow-gauge track where z = 4 inches and x = .67 ft.

Then t ± x + z = .67′ + .33′ = 1 foot.

Substituting this value of t in formula (4) $l = \sqrt{2 \times 1 \times 478.34}$ = 30.9 feet, = distance of head-block from point of curve. The practical length of the switch-rail is ¾ of 30.9 = 23.1; therefore it is to be spiked ¼ of 30.9 from the P. C. towards the head-block, or as much as is necessary for good line.

Increase of Gauge.

In turn-out curves, with a radius of less than 1600 feet, it is necessary to widen the gauge. The increase of gauge required for any radius is determined by the following formula from Molesworth, p. 201.

Let R = radius of the curve in feet; y = increase to be given to gauge in inches. Then $y = \frac{562.5}{R}$ . . . . . (7).

In no case is the increase to exceed ⅞ inch.

In applying this formula to the case shown in Fig. 7, which will be the same for any other case, proceed as follows:

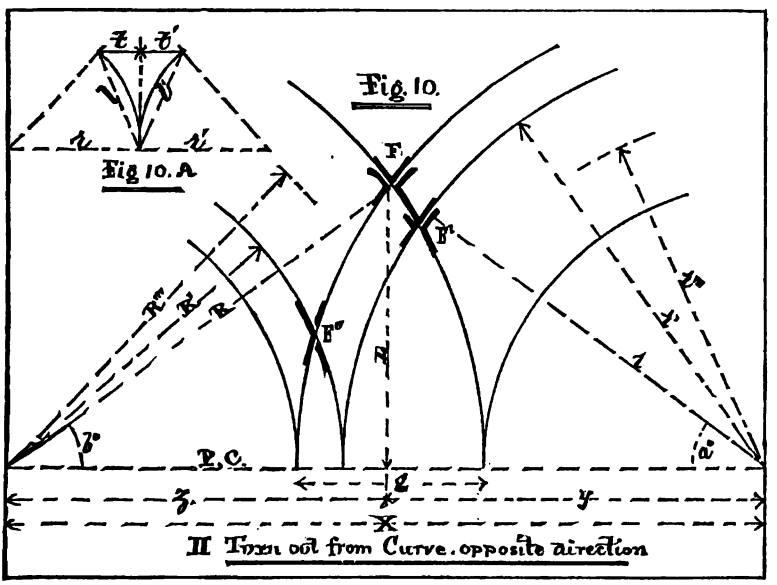
$$y = \frac{562.5}{478.34} = 1 + \text{in. use } \tfrac{7}{8} \text{ in.} = .07 \text{ ft.}$$

Now, by shifting the two right-hand rails of the turn-out .07′ further to the right, the gauges will be widened the proper amount without disturbing the position of any of the other rails or frogs, except frog b. The radius of the rail crossing frog b = 477.51. Taking .07′ from this, we have for the new radius, 477.44 feet; we can then proceed to find the new lead and frog angle for frog b, but this is hardly necessary, as the difference will not be appreciable. The best mode of procedure, after having all the frogs in position and line except frog b, and having all the lead rails curved the required amount, and in exact position, is then to place the frog b in the position as calculated, and in line with main track.

Then use a gauge and measure the true gauge + the increase required from the rail crossing the frogs c and e, over to the frog b and perpendicular to it. Then by sliding the frog b towards the H. B. a trifle, after one or two trials, keeping frog b on line with main track at the same time, we find a position for b where the gauge is widened the proper amount. The outer right-hand rail is then also measured in with the increase of gauge required, and spiked.

II. TURNOUTS FROM CURVES IN OPPOSITE DIRECTION.

Taking the case of the combination of two gauges with three rails, which will illustrate the method of solving the other cases of



combination track, by reference to Fig. 10 we obtain the following results:

R''' and r''' = radii of centre lines of the two curves; then taking the case of frog F as an example, proceed as follows:

R and r = radii of the two turn-out rails intersecting at the point of frog F.

g = as before, the perpendicular distance between these two turn-out rails under consideration at the P. C. Then $x = (R + r) - G$.

$$z - y = \frac{R^2 - r^2}{x} = 0 \text{ if } R = r.$$

$$y = \frac{x - (z - y)}{2} = \frac{x}{2} \text{ if } R = r$$

$$z = x - y = y \text{ if } R = r.$$

$$L = \sqrt{r^2 - y^2} = \sqrt{R^2 - z^2} \quad \ldots \ldots \ldots \ldots \quad (8).$$

$$\sin a^\circ = \frac{L}{r} \quad \sin b^\circ = \frac{L}{R}$$

$$\text{frog angle} = a^\circ + b^\circ \quad \ldots \ldots \ldots \ldots \quad (9).$$

Similarly, if we let fall perpendiculars from the frogs F′ F″ we find new values for x, y, and z, then drawing the radii to their points, we obtain other values, to be used in determining the leads, etc., for the other frogs, as follows:

For frog F′ use r′, R, a′°, b′°, L′ z′ Y′ g′.
" " F″ " r, R′, a″°, b″° L″ z″ Y″ g″.

Apply formula (3) for finding the frog numbers.

A three-throw switch for combination track may be solved by the combination of the above with the problem of the turn-out from straight track, Case 1. Generally it is bad practice to use a three-throw switch in these cases. It is best to have a second side-track turn-out from the first side track, or from the main line at some other point.

To find the length of the switch-rail in the case of Fig. 10, we see that the rail is to have a curve when thrown either to R or L. That in the case of a 5-inch throw when R = r, the deflection of the switch-rail at the head-block on either side of a tangent to the P.C. is 2½ inches. Therefore, substitute 2½″ for t in formula (4) for the length of switch-rail when R = r. If R is not equal to r, proceed as follows:

Let t and t′ = throws to R and L of the tangent l and l′ = length of switch-rail to point where sum of throws = 5 inches; r and r′ = radii of the two curves.

$$t = \frac{l^2}{2r} \quad t' = \frac{l'^2}{2r'}$$

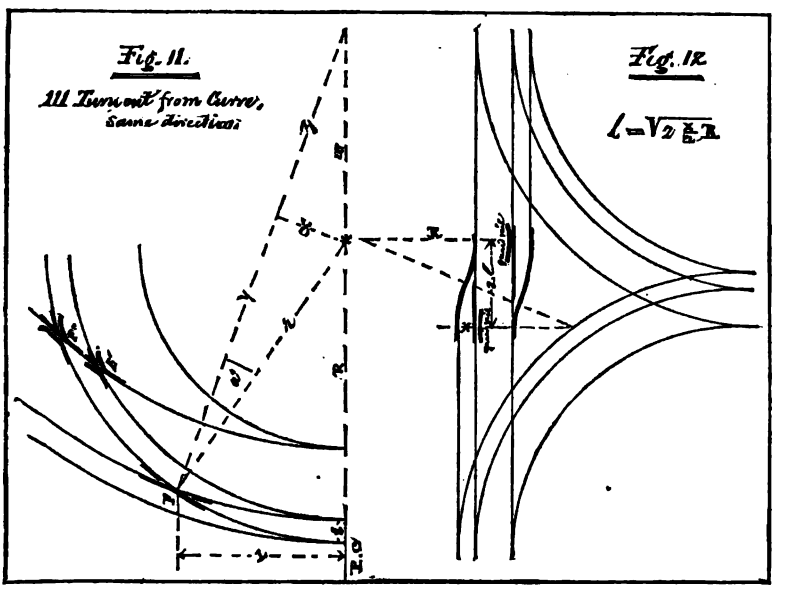
$$t + t' = \text{total throw } T = \frac{l^2}{2r} + \frac{l'^2}{2r'}, \; l = l' \text{ approx.}$$

$$\therefore l^2 = \frac{2rr'T}{r + r'} \quad \ldots \ldots \ldots \ldots \quad (10).$$

It may be necessary to spike the switch-rail ¼ of the way from the P. C. up, and drive spikes or lugs in the ties to hold the rail in a curve by pressing against them when thrown for either track.

III. Turnouts from Curves in Same Direction.

Knowing the radii of the centre lines of the curves, we can find the radius of any one of the rails. Taking the case of frog F (Fig. 11), for example, let R and r denote the radii of the two rails inter-



secting at the point of frog F; g = the perpendicular distance between them at the P. C.

then $w = g + (R - r)$

$$y - z = \frac{r^2 - w^2}{R}; \quad y = \frac{R + (y - z)}{2}; \quad x = \sqrt{r^2 - y^2}.$$

$$L = \frac{Rx}{w} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (11).$$

$$\text{Sin } a^\circ = \frac{x}{r} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \quad (12).$$

$$\text{if } w > r,\ z - y = \frac{w^2 - r^2}{R};\ z = \frac{R + (z-y)}{2};\ y = R - z$$

Similarly we can find the leads and frog angles for frogs F′ and F″ by repeating the same construction and notation as shown in Fig. 11 for frogs F′ and F″. The other cases of combination track are solved in the same way as the above.

Apply formula (3) for finding the frog numbers.

The switch-rail may be curved or set by force for the curve with the greater radius, then calculate the length of the switch-rail, using the small radius and given throw, by formula (4). Follow directions as for former switch-rails.

The second case of combination track is used, for example, where cars of different gauges are transferred across a river on boats, and where, for certain reasons, a combination of three rails will not answer. Especially where boats receive and land the cars at the same end or where the transfer is made between three or more points along the river.

Four rails are also used where it is required that cars of different gauges, carrying ore, coal or coke, etc., shall dump the material evenly on both sides of the track.

The third case of combination track occurs where the difference in the gauges is small, as at Cincinnati and other points where Southern roads (gauge about 5 feet) connect with standard-gauge roads; also where the gauges are the same, but each road uses its own rails.

In all these cases of combination track it is only necessary to run centres for one of the gauges; and knowing the eccentricity of the other gauge, the distance of all the rails can be determined with reference to the centres run.

A method of finding the leads of the points of frogs with an instrument on the ground, is to set up over the P. C. of the turnout over the centre line. Say we want to find the position of frog F″ (Fig. 5). We know that the point of frog comes at the point where the centre line of the turnout deflects from the centre line of the main track by an amount = to the distance between the gauge side of the two left-hand rails (= 1.7); so that by trying the deflection with the instrument for a certain distance, and measuring the deflection from the tangent for such a degree of curve, and if too small try the deflection again with longer distance, or if too large, try deflection again with smaller distance, and repeat till you find a place where the deflection = the distance between the gauge side of the rails.

For frog F′ the deflection corresponding to the required lead is 3 feet; for F it is 4′ 8½″. Having found the lead, and knowing the degree of the curve, the frog angle can be found by approximation as follows:

$$\text{Frog angle} = \frac{\text{degree of curve} \times \text{lead}}{100} \text{ or frog number} = \frac{R}{L}.$$

The crossings must be laid in rails of the same weight. In the cases 2 and 3 of combination track, as soon as the crossings are passed, rails of different weight can be used, but in case 1 the rail of the heavier weight is used till the combination ceases.

Fig. 12 shows a method to be used in a "Y" where the three sides are laid in combination gauges of three rails. The above is best adapted for narrow-gauge roads with sharp curvature.

IMPROVISED SURVEYING INSTRUMENTS.

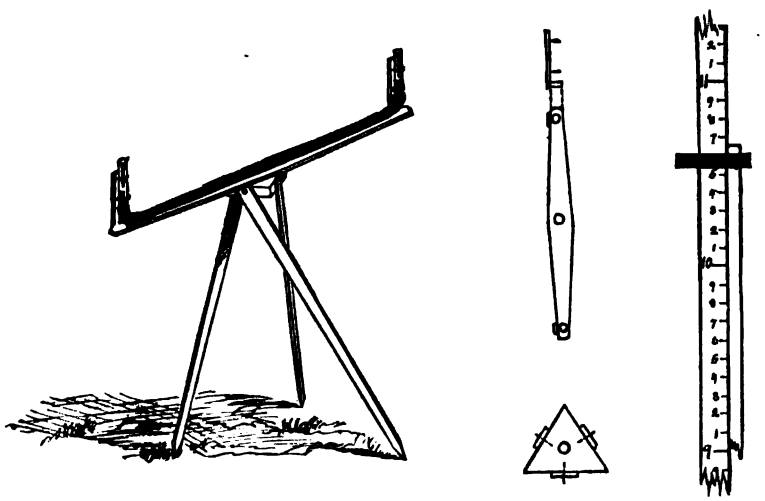
BY JAMES L. GREENLEAF, C.E.

STUDENTS of engineering will frequently have the opportunity of employing some of their vacation usefully to others, if not profitably for themselves, in making surveys for small improvements, such as are constantly occurring in country districts and about villages. The changing of a highway that has been decided upon by a town committee, the grading of a sidewalk, the laying out of grounds around a country home, the leveling, and drawing up of plans for drainage of a marsh, and many other improvements not requiring years of experience and skill, are well worthy of the ambition of a man who aspires to the aligning of trunk lines instead of a country road, or hopes that the small retaining wall he has staked out may be only the shadow of great marine works to be designed by him in the future.

A person with a little engineering knowledge and some instinctive sense of taste within him, can go into a country place during the summer months and in one way and another apply his powers, and he will be surprised to see in the course of a few years what an influence he has had upon the neighboring farmers, or the town officers. They will begin to realize that it is sometimes cheaper

to grade a road at the first, even if some cutting and filling be necessary, than for generations of horses and human beings to lift themselves day after day up the pitch, only that they may drop down the other side. They will gradually smile less with good-humored compassion upon "that city fellar" applying the laws of the spiral or circle, and curves of more grace will begin to take the place of those which obey a law all their own, and which only a farmer can lay out in full perfection.

Such is the opportunity which a young engineer often has of applying his knowledge, obtaining some experience, and at the same time of giving valuable assistance to his neighbors or the community in which he stays. The difficulty that occurs at once is that he has no instruments. The desire in this little article is to indicate how he can, with the expense of a dollar or so, make such as will serve him a most excellent purpose. The writer has had occasion to make and use them in a variety of work when no regular surveying instruments were accessible, and has been surprised and gratified at the comparatively accurate results obtainable.



The taking of levels, the measurement of angles, and the measurement of horizontal distances are the usual operations of surveys.

The direct measurement of angles, to be of any value, requires instruments of precision, but this difficulty can be avoided by using the plane-table method.

Very fair results can be obtained by using a drawing-board for

a plane-table and mounting it upon a tripod such as will be described for the level. For the alidade can be substituted a simple straight-edge with a needle stuck vertically close to the edge, near each end. The straight-edge should be divided into inches and tenths, and smaller divisions can be made by eye. The drawing board is mounted upon the tripod plate with an ordinary carriage bolt. This bolt is countersunk in a piece of board 5″ or 6″ square, which is then screwed fast to the under side of the drawing-board. A little care in setting the tripod will make the table practically level.

A linen tape can be substituted for the surveyor's chain, but it is a question if it is not a better way to take a hemp (not cotton) clothes line and tie rags *through the strands* for the even tens of feet; with strings instead of rags for the other divisions. The number of feet can be marked on the cloth with ink.

A water level will give very satisfactory results if used to determine levels, and can be readily constructed. Such a one as is illustrated requires a yard of rubber tubing, two 10-inch lengths of ½″ glass tube, four screw eyes, two corks, a carriage bolt, and some strips of wood.

The illustration hardly requires description. The level bar is a strip 2½′ to 3′ long, and 3″ wide at the middle. At the side near each end, is nailed a light upright some 6″ or 7″ long, into which the screw-eyes are screwed. The two glass tubes are held by these latter, and their lower ends are connected by the rubber tubing. A carriage bolt through the middle of the bar serves to hold it on the tripod. It is well to omit the nut on the bolt, as then the level can be easily detached from the tripod and carried separately, A little ink mixed in the water will color it sufficiently.

The tripod consists of an equilateral triangle of, say, 5″ on the side, cut out of 1¼″ or 1½″ plank, with a leg hung to each side by a 2″ screw. A hole for the bolt is bored in the centre of the tripod plate.

For the leveling rod a light 15′ strip can be obtained from a carpenter and divided into feet and tenths. These tenths bisected will enable one to estimate the hundredths of a foot quite accurately.

A target can be made by nailing a cross-bar to a strip about 7′ long, which is held on the side of the rod and slid up or down.

For a low sight the target can be reversed, with the cross-bar downward. If one side of the cross-bar is colored black with ink or paint, and the other side left white, a contrast can always be obtained with the background, according as it may be light or dark.

With these simple instruments much satisfactory work can be accomplished. The list is by no means exhausted, and possibly improvements can be suggested on those described, but experience has shown that for a great many cases, practically as serviceable results can be obtained with them as when more exact and more expensive intruments are used. That this paper may convey a few useful hints to the engineering student is the desire of the one who has written it.

THE LURAY CAVERNS.

BY GRAHAM LUSK.

THE stalagmitic caverns of Luray were first discovered by Andrew J. Campbell on the 15th of August, 1878. Campbell was one of a party of search organized by Mr. B. P. Stebbins, a travelling photographer, who was convinced that a cavern existed somewhere in the vicinity. A small cave in one of the hills near Luray had long been known, but the discovery of Campbell was a great surprise to the villagers, and it brought to light one of the finest pieces of subterranean architecture in the world.

Luray is situated rather to the north of Central Virginia, in Page County. The surrounding country is a broad valley of rolling limestone, bounded on the east by the Blue Ridge and on the west by the Massanutton Mountains. The cavern is about a mile from the town, is in the middle of the valley, and is about five miles distant from either of the bordering mountain ranges. It occupies a rounded mound locally known as Cave Hill.

The Luray Cave has been hollowed out of Lower Silurian limestone, probably during the tertiary period, but possibly at an earlier date. First it consisted, perhaps, of a slight fissure with insignificant ramifications, through which flowed water charged with carbonic acid. Little by little the limestone was dissolved, the insoluble carbonate changing to the soluble bicarbonate, and being

carried away by the stream. Another active agent in the excavation was the sand held in suspension by the water, which, simultaneously with the solution of the limestone, wore away the sides of the channel by erosion. In the course of time, as the cavern increased in size, the process of excavation was accelerated by the falling of large masses of rock from the roof. After having been the course of drainage for countless years, the channel of the flowing water was altered by some geological change, thus leaving empty, vast halls stretching in every direction. The process of excavation being completed, the secondary process of ornamentation now set in. Rain water always contains a considerable quantity of carbon dioxide dissolved from the atmosphere, and this amount is increased when the water comes in contact with the decaying vegetable matter at the surface of the ground. Now when water, charged in this way with carbonic acid, percolates through limestone rock, the limestone is dissolved and held in solution as a bicarbonate. Let this aqueous solution emerge drop by drop on the ceiling of a limestone cavern; coming in contact with the air it immediately loses a part of its carbonic acid, thereby depositing a minute amount of insoluble limestone which adheres to the ceiling; the drop hangs a moment and then falls to the floor, still charged with a little of the soluble bicarbonate. By the time it has reached the floor it has lost all, or nearly all the carbonic acid it contained, and the insoluble limestone takes its place on the bottom of the cavern, to contribute its infinitesimal quota to the formation of a mighty column. Sometimes the water emerges through the ceiling following certain lines, and in this way are formed the beautiful draperies of translucent calcite, which astonish one by the gracefulness of their natural foldings. The Luray cavern, after its excavation, was subjected to this process of adornment, and many large pillars were formed before it again became a drainage channel, this time for the corrosive mud of the glacial period. The mud darkened the whole surface of the cave, and carved out numerous contortions. When the ice and glacial mud were things of the past, a second growth of stalactitic and stalagmitic forms arose from the dark-brown walls of the cave; many of these formations are of the purest white, while others are colored with oxide of iron. A unique feature of the cave is the presence of *helictites*, which are lateral outputs from the sides of the cave and from the pillars; the cause of these curious grotesques has never

been explained. They often surround the columns of the cavern in several distinctly separate growths. Washington's Pillar is highly fluted in this manner, and in the Fish Market these helictites hang over the rock as a fringe, curving so as to remind one of elongated alligator's teeth.

This brief review of the formation of the cave will make more interesting the description of its most prominent beauties. The caverns are illuminated with the arc electric light, and in this way they are seen to best advantage, though the sense of vastness is thereby much decreased. When a couple of candles struggle to dissipate the gloom the dimensions of the halls seem to be almost limitless. When the cave was first discovered numerous tracks and indentations were found in the clay of the floor, thus proving it to have been the former abode of wild animals. A part of a human skeleton may be seen imprisoned in that portion of the cave which bears the ghastly name of Skeleton Gulch. The entombed skeleton is of no scientific interest, as its confinement in the tufa of the floor evidently took place at no very distant period. At one portion of the cave we reach a place where the ceiling comes within a foot of touching the floor; here the pathway has been dug out to a depth of three feet.

Hence, we enter a spacious chamber which contains Pluto's Chasm, a deep cleft in the rock, 500 feet long, 70 feet high, and of a width varying from 10 to 50 feet. From the deepest and gloomiest part of the chasm rises a pure white stalagmite called the Spectre, which for centuries has haunted the locality with its ghostly appearance. One of the halls is called the Cathedral, and contains a wonderful Stone Organ. This latter consists of parallel blades of stalactites, which are beautifully sonorous when struck. Each blade seems to give forth a different note, and tunes are played upon them, the blades vibrating in low, clear, harmonious tones. Near by are the Chimes, and here again we have evidence of the sonant power of this stalactitic calcite. The Giant's Hall is the culminating triumph of nature's handiwork at Luray. It is a chamber 80 feet in height, and among the multitude of beautiful pillars the Empress Column at once attracts attention. This is a very slightly rose-colored stalagmite reaching to the height of 35 feet, and having a most beautiful symmetry. Near by is the Henry Baird Column, an immense bulwark of snowy calcite. There are many other large columns in the Giant's Hall, and

besides all these, the ceiling is an inverted forest of smaller stalactites. There is also the Cascade, a pure white formation resembling a foaming torrent pouring out of the rock. Several of these beautiful cascades are found in the cave, and they are among its finest adornments. The Angel's Wing is an exquisite formation of the purest white, and resembles a wing as closely as if sculptured by a human hand. Many of these names are manufactured to aid the guide in telling his story, but others are wonderfully appropriate. At one point we are shown a Castle on the Rhine, which resembles Bishop Hatto's Mouse Tower. Again our attention is drawn to a large owl perched upon the side of a stalagmite. One of the marvels of Luray is the Hollow Column, which, in former years, has been the course of a stream of water; this column may be ascended by means of a rope, and 60 feet above the floor, another large chamber is opened to the enterprising visitor.

There is no considerable sheet of water at Luray, but there are several pools of various sizes in which the water is as clear as glass. From these basins crystals of calcite are being deposited by the slow evaporation of the water. While one is walking through the great halls of the cave, he is almost certain to turn his eyes upward to the long pointed stalactites weighing hundreds of tons, and the story is apt to come to his mind how a certain king's subject, in the gratification of his own desire, was permitted to take the king's place at table, and how, when he had half finished the meal, he looked up and saw a sword hanging by a single hair. The cohesion, however, of calcite is very great, and a fallen column is of rare occurrence.

The fauna of the Luray Cavern is very limited. There are numerous spiders, all of which belong to the species *Linyphia weyeri.* There are a few *Spirostrephon copei*, which differ from those found in the Mammoth Cave, in having shorter hairs. A myriapod, called the *Syngonopus whitei*, has also been discovered. The fluffy, white growth of mould on the green planks of the walks seems to be the only form of vegetable life in the cave.

In the comparison of Luray with the great caverns of the world, it may safely be stated that no other is so completely decorated with stalagmitic formations. The Mammoth Cave consists of an almost endless series of chambers stretching underground for miles, while the Luray Cavern has less than two miles of avenues; but in the former cave the halls are vast darkened chambers,

whereas, at Luray they are delightfully varied and embellished. The finest limestone cavern in Europe is at Adelsberg, in Austria, and this is laid out in walks four to five miles long; Adelsberg is somewhat similar to Luray, and there is one beauty which the latter cavern does not possess. This is the presence of dry stalagmites and stalactites, which are covered with crystalline faces, and which glisten as though illumined with myriads of diamonds, even with the feeble light given by candles. At Luray there is no flowing water; at the Mammoth Cave some of the halls are still the beds of a subterranean river, while at Adelsberg a deep river, strange to say, enters the mouth of the cavern, falls downward, and disappears, to be heard of no more.

There is something weird and fantastic in excursions through the chambers of solid rock in these underground caverns, shut off, as they are, from all the outside world; it is a feeling of strangeness and wonder, which, after one's visit, becomes a pleasant memory of hours delightfully employed.

UNEDUCATED REASON IN THE CICADA.

BY J. S. NEWBERRY.

In 1877, a colony of the seventeen-year locusts (*Cicada septen. decim.*) appeared at Rahway, N. J. During the interval between the appearance of that and the preceding generation, the town had been extended, and some houses had been erected where forests or fields existed before.

One of these houses—that belonging to Mr. Alonzo Jaques—was constructed on the site of an old orchard, and had a shallow cellar. This cellar was kept closed till about the time of the advent of the cicadas; the door was then opened, and the bottom of the cellar was found to be thickly set with mud-cones or tubes, from six to eight inches high, an inch to an inch and a half in diameter, each of which had been formed by the pupa of a cicada, that had emerged from the earth beneath the cellar. Finding a dark chamber, and apparently desiring to work up to daylight, the cicadas had taken the moist clay, and of this formed pellets, with

TUBE FORMED BY PUPA OF CICADA.

which the tubes were built up, apparently with the purpose of bridging over the vacancy and thus reaching the surface.

These facts appeared to me so interesting that I procured a large number of the tubes, one of which is here figured, and I had the first report verified by the written testimony of the owner of the house and several other well-known citizens of Rahway.

The document sent me with the tubes has remained in my possession to the present time. It is dated June, 1877, and reads as follows:

"These cones were erected by the pupas of the cicada in the cellar of a house belonging to Alonzo Jaques, Rahway, during parts of May and June, 1877. They were built in an unfloored cellar of a house constructed about eight years ago in an old orchard. The cellar was dug to about the depth of a foot in red clay, and the bottom covered by a slight layer of débris, sand, sticks, etc. The cellar was perfectly dark during the construction of the cones, the only opening being shut. The locality is a dry one, the house being situated on a rise of ground, and about a quarter of a mile from the nearest water—a ditch dry in summer. These cones were not seen in the course of erection, but when the cellar was opened, about the time the locusts made their first appearance, the whole cellar bottom was covered by them. The tops of all were closed, but on breaking some of them the pupas were seen both in the hole in the ground and in the cone.

"After the cellar had been opened and left so, they appear to have stopped building and to have made holes in the tops of the cones for their exit. These cones were a great curiosity to the people of Rahway, and many came to see them, declaring them something entirely new in their experience.

(Signed) A. E. Crow,
Alonzo Jaques,
W. B. Devrie,
M. L. Crow."

In the facts cited above we have evidence of the exercise of intelligence in the cicada, and a judicious adaptation of means to an end in circumstances that it would seem must have been without precedent in the experience of that or any preceding generation; and, therefore, for which no education of ancestors could have given a preparation. It is possible that the pupa of the cicada

is sometimes embarrassed in its ascent to the surface, by water, by too wet or too dry sand or mud, but it is hardly possible to imagine circumstances where the construction of a tunnel would be necessary.

In the earth, caves of any considerable size rarely or never occur, since surface water is constantly flowing through all superficial materials, and filling cavities with transported matter. Caves often occur in rocks, but the cicada has no power to penetrate rock, and lives in earth near the surface.

Perhaps some of those who have made the habits of the cicada a study, can suggest a school in which they could have received the training that fitted them for the engineering work they attempted in the case under consideration; yet, though I have studied the habits of various colonies of the cicada with some attention, I am quite at a loss for any explanation of the phenomena that will bring them within the scope of the theory according to which all our organs and faculties are the result of formative influences progressively developed through a long line of ancestry.

In whatever way the problem shall be solved, it has seemed to me of sufficient interest to warrant placing the facts on record.

THE PYRITES DEPOSITS OF THE ALLEGHANIES.

BY ARTHUR F. WENDT.

THE ancient slates, forming a considerable portion of the earth's crust, have, from time immemorial, been cited as storehouses of great mineral wealth. Second only to the vast iron-ore deposits within their bounds, are the huge bodies of iron-pyrites found imbedded therein, in every quarter of the globe.

America, rich in all minerals, has not been treated niggardly. From Alabama to Newfoundland, the slates of the Appalachian range contain, at intervals, enormous beds of pyrites. Copper, nickel, cobalt, zinc, lead and silver are constant attendants of the ore beds, but are found only as minor ingredients in the mass of the pyrites, the "mundic" of the miner. Even now, the small percentage of copper, contained in the mass of the iron-sulphurets, has become a source of copper, and promises to become (as in

Spain) a large future source of the metal and one that will eventually exert a very decided influence on its commercial production and value in America.

The country rock, in the immediate contact with the beds of pyrites, is usually talcose or quartzose sometimes argillaceous or chloritic slate, changing by degrees into mica schist and gneiss rock.

Staurolites and garnets, in large quantity, are frequent in the schists. Fine crystals of hornblende and epidote are not rare, and calcite, quartz, mispickel and molybdenite are not unusual. Geologically, the ores are found in the Taconic or Eozoic and Silurian slates, although occasionally more modern formations, for instance, the Devonian slates of the Hartz Mountains, serve as their repository.

In every instance the pyrites form true ore-beds (Erz-Lager), and, as such, present evidence of contemporaneous origin with the inclosing rocks, or, more properly speaking, of deposition subsequent to the original foot-wall, and anterior to the original hanging-wall. It is, indeed, often noted that the character of the slate-walls of these pyrites deposits differ widely, showing a change in the conditions of the deposition before and subsequent to the accumulation of the pyrites. The source of these pyrites has been and still is a vexed question. Some noted geologists (T. Sterry Hunt amongst others) have held that some of them are fissure veins.

To my mind, such conclusion, even were it admissible, only increases the difficulty. It is scarcely credible that fissures, miles in length, and in places hundreds of feet in width, could exist a sufficient time to allow of their being filled uniformly with a deposit of pyrites. Sharply defined horses of country rock would, certainly, be frequent, whereas, in fact, they do not, to my knowledge, occur in any pyrites deposits.

It is much easier to conceive of a deposit of pyrites accumulating in a manner similar to an iron-ore deposit, and of a subsequent concentration or mechanical rearrangement of the unsolidified grains of such a deposit into basins or hollows, thus eventually forming the enormous characteristic lenticular masses of pyrites found at widely distant points, of which the following are the most noted: Ducktown, Tennessee; Ore Knob, North Carolina; Louisa County, Virginia; Vershire, Vermont; Charlemont, Massachusetts; Notre Dame Bay, Newfoundland; Fahlun in Sweden; Rammelsberg in the Hartz Mountains; Agordo in the Venetian Alps; the provinces

of Huelva in Spain, and Alemtejo in Southern Portugal. All these localities bear witness to the same forces displaying the same activity, and with the same results, at almost the same geological epoch.

The great width of these deposits, reaching 350 feet at Ducktown, 250 feet at Agordo, 400 feet at Fahlun, 320 feet at Rio Tinto (Huelva), and the even greater lengths varying from 600 to 3000 feet, and, furthermore, their remarkably uniform character throughout, would alone render a vein formation most improbable.

Neither can a vein formation or origin explain the occurrence of what, at Ducktown, is termed "slate ore." That is a true slate in no way differing from the country rock, which contains copper pyrites in minute lenticular masses, so dovetailed and inter-stratified, and following every flexure of the slate itself, that a contemporaneous origin must, of necessity, be accorded to it.

FIG. 1.

Pyrites deposits of Rõoras.

At Ducktown, the Hiwassee series of mines, forming a belt several miles in length, are thus constituted. In Fannin County, Georgia, a similar occurrence takes place. At Stone Hill, Cleburne County, Alabama, R. P. Rothwell has described a like deposit. Ore Knob, although more pyritous, will undoubtedly fall within the category of ore-beds; and further north the Virginia mines, the Gap mine in Lancaster County, Pennsylvania; St. Anthony's Nose, New York; Charlemont, Massachusetts, and many of the deposits scattered through New Hampshire, Vermont, Maine (notably at Blue Hill), and at Capelton, Canada, and in Newfoundland, present similar features, and *all* must be classed as ore-beds proper.

Von Cotta describes a similar bed of "slate-ore," many miles in length, as occurring in the Carpathian Alps in the Bukowina of Hungary; Von Groddeck, a similar quartzose bed at Oryarfvi, Finland.

Röoras, in Norway, presents a striking example of the formation of these pyrites beds with very little subsequent change of position. Two beds, each 10 feet thick, are superimposed on each other. The cut clearly illustrates their character (Antwerp Exhibition). The lower bed is 1700 metres long from cropping to cropping; the upper, 500 metres. In a north and south direction the lower bed has been worked over an area 600 metres wide. The hill of hard Gabbro has, evidently, protected the underlying measures from erosion and preserved the pyrites beds intact.

In one of the late numbers of the *Zeitschrift für Berg-, Hütten- und Salinen-Wesen*, Albert Nettekoven describes these mines and pronounces them fissure-veins. Like many other geologists he errs in adducing pitch of vein on the strike, smooth walls, selvages, etc., as proofs of the character of the ore-bed. Such walls and selvages are, it is true, ALWAYS present in some parts of a fissure, but they are not peculiar to them alone. A perusal of this paper will note their occurrence in ore-deposits of undoubted character, to wit, the magnetic iron ore-beds of the Appalachian range.

As regards Ore Knob, it must have been held to be a vein, largely from the fact of its exceptional length and comparative small and uniform width. But numerous similar instances can be cited amongst iron deposits, which all geologists admit to be ore, beds proper. Thus, the Chateaugay mines of Lake Champlain, the Croton mines of New York, and the Hibernia mines of New Jersey are characteristic iron-ore bodies of this class, while the Port Henry, Tilly Foster, Mahopac, Forest of Dean, all in New York State, and Dickerson, of New Jersey, approach the lenticular shape, and are wide and short.

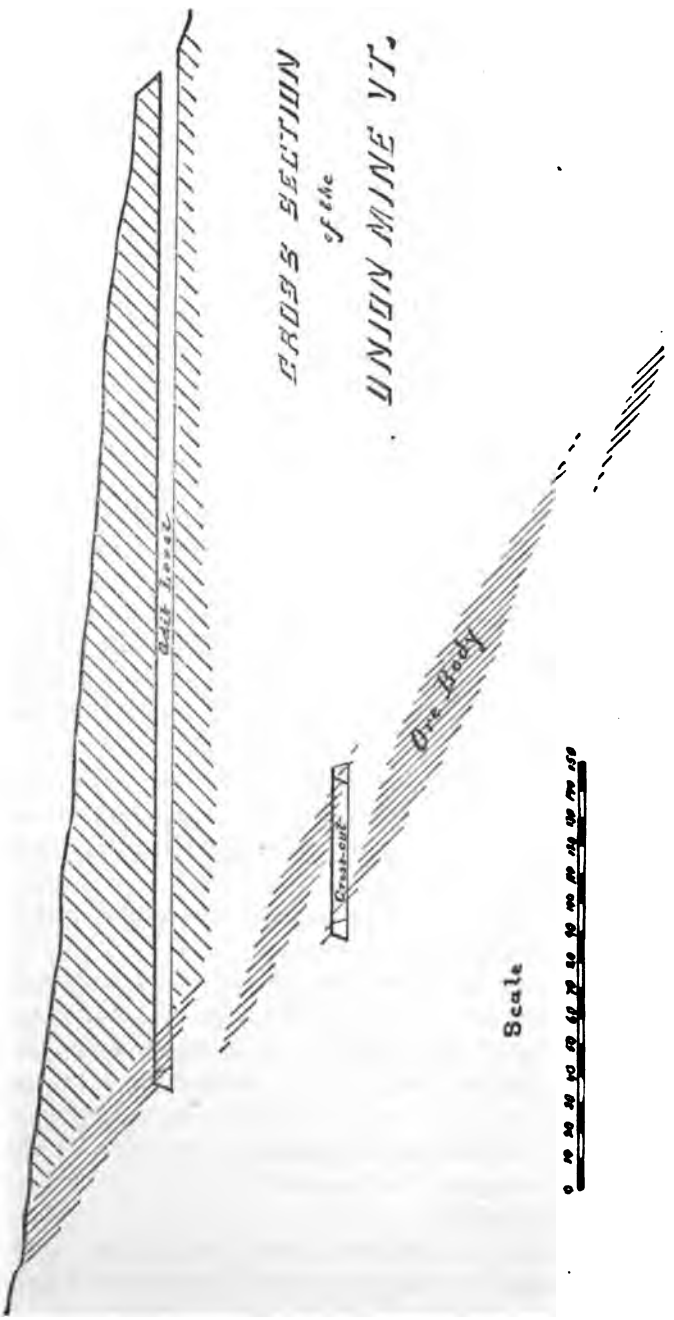
Indeed the occurrence of iron ores and iron-pyrites in the Alleghanies have many points in common.

Sometimes these beds have smooth well-defined walls, even a gouge is found in some instances. The Tilly Foster and Mahopac iron mines and the Ore Knob and Mary (Ducktown) mines are cases in point. At other times the ore is scattered through the country rock, and the richness in metal only limits the operations. The Chateaugay iron-ore belt and Hiwassee pyrites belt are instances.

Often both phenomena can be observed in the same ore belt, or even in the same ore bed.

The occurrence of smooth, well-defined walls cannot be surprising if the dissimilarity of the ore and inclosing rock is considered.

FIG. 3.


CROSS SECTION
of the
UNION MINE VT.
Calf Tunnel
Cross-cut
Ore Body
Scale
0 10 20 30 40 50 60 70 80 90 100 110 120 130 140 150

The line of junction of such dissimilar material would naturally become a well-defined plane of motion in the metamorphism and subsequent elevation of the strata, thus producing the slickensides and gouge commonly attributed only to true fissure-veins. Perhaps the most conclusive proof of the character of these deposits has been given by the Fahlun, Agordo, and East Tennessee mines. In depth they all pinch and eventually are completely cut out.

At Agordo the ore body disappears at a depth of 500 feet. At Fahlun, the section of the ore at the 50-fathom level is 110 × 120 × 140 × 160, and these dimensions diminish to 50 × 40 × 50 × 43 feet at the 180-fathom level. Within the last few years a huge cave has taken place at Fahlun, by reason of the almost complete exhaustion of the ore-body and consequent attempt to remove pillars. A most striking example is furnished nearer at home, by the East Tennessee mine, of Ducktown, of which a sectional plan is annexed. This ore deposit has been sometimes erroneously pronounced a fissure-vein. The section shows the complete pinching out of the original ore-body and the cutting of what is undoubtedly the edge of a similar one in depth. This new discovery naturally leads to the question of the persistence of these mines in depth. (Plate I.)

Taken in the usually accepted sense of uninterrupted continuance as in a true fissure-vein, these deposits certainly come to an end in depth.

For all practical purposes, however, they extend indefinitely in depth, as it is undoubtedly a general law that on the pinching of one ore-body a new ore-body will be struck by sinking lower and cross-cutting into the foot. This has been repeatedly the case at Vershire, Vt., and is known to have occurred in many iron mines, both in the East and Lake Superior, and in the similar Franklinite mines of New Jersey.

So strongly has this fact been impressed on the writer that explorations, conducted by his advice, on the basis of this law amongst others, at the Union mine, Vt. (see Fig. 3), have led to the discovery of fine bodies of rich ore in a practically barren and abandoned mine.

Prospecting under similar conditions will surely lead to similar results elsewhere.

The ore-bodies in the old slates are strung along the strike of the rock as peas in a pod, or sometimes "en echelon." On their strike, the ore-bearing series often measure many miles, and there

is no valid reason why the same agencies that have produced these lenses of ore along the present croppings of the rocks should not have acted in depth as well.

If we consider that what are now the croppings are so simply by reason of extensive denudation, and that these croppings formerly constituted the rocks at great depth below the surface, it would seem a self-evident law that so long as the slates continue towards the deep, bodies of pyrites may be confidently searched for.

The croppings of the beds of pyrites are always more or less decomposed, depending primarily on the *latitude* under which they are found.

All the northern beds of pyrites have had their decomposed portions in a measure scoured away by glacial action, and crop out almost unchanged as yellow sulphuret.

Such was the case in Canada and Vermont. Probably also in Norway and Sweden, for the occurrence of oxides and carbonates is not mentioned at Fahlun and Rōoras.

But in Spain and in Virginia, North Carolina, Tennessee, and Georgia, the pyrites have been completely decomposed to a considerable depth, sometimes several hundred feet, and in their place huge masses of iron croppings, the "gossan" of the Cornish miner, cover the surface.

Below the gossan and *above* the undecomposed yellow sulphurets, the copper that was originally contained in the croppings has been reprecipitated, forming the rich layers of "black" ore, which first gave fame to the Ducktown mines and which were worked many centuries ago by the Romans in Spain.

Two sections of Ducktown mines are annexed which clearly illustrate the occurrence of these decomposed ores at the Mary and Isabella mines.

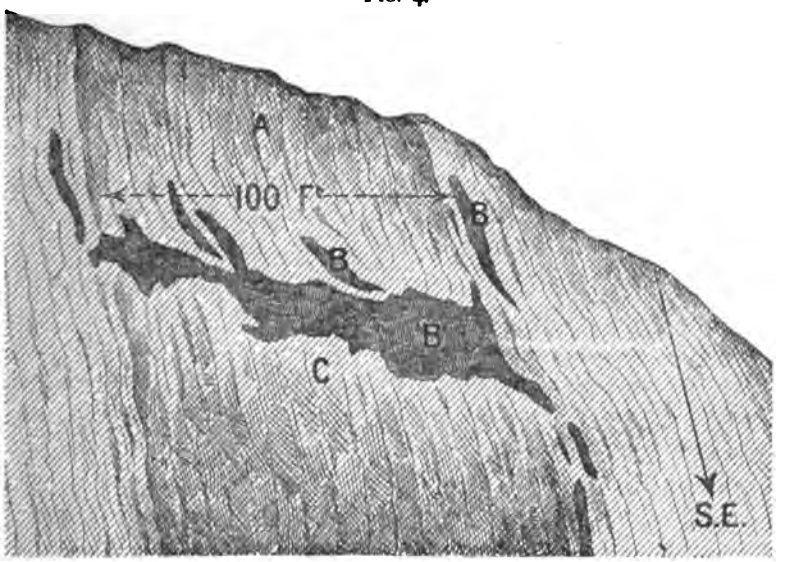
That this decomposition is still steadily going on is evinced by the copper waters pumped from some of the mines.

Thus the Rio Tinto (Tinted River), of Spain, owes its name to the products of decomposition of the Huelva mines carried off by its waters.

Over forty years ago when the Ducktown mines were first opened, the drainage from them entering the Ocoee River (a rapid-flowing river over one hundred yards in width), vitiated that stream to such an extent that fish died by tens of thousands, creating a veritable pest house by their decomposition along its banks. Even now

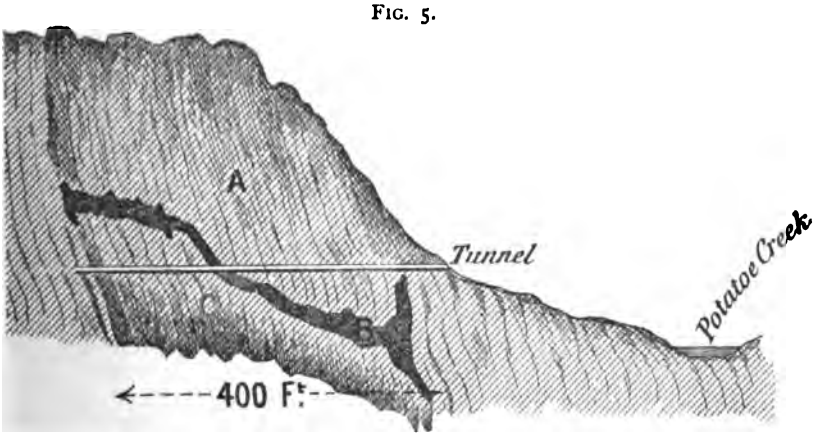
the strong chemical action going on underground is evinced by the heat of the ground over the Eureka mine, of Ducktown. Snow

FIG. 4.



will barely lay on the ground above the mine before melting, and a wooden pump, raising the mine water to the surface, is all the

FIG. 5.



A—decomposed ore or gossan.
B—rich oxidized ores.
C—decomposed yellow pyrites.

machinery required to obtain 3000 pounds of cement copper per month from the stagnant mine waters.

In 1859, the Hiwassee, Eureka, and Isabella mines produced 40,000 pounds of copper monthly from this source alone.

The history of all the pyrites deposits of the Appalachian chain is best illustrated by that of Ducktown, the first to be discovered, and the one wrought on the largest scale commensurate with its resources.

The Ducktown copper district is located in the southeastern corner of Tennessee, and but a few miles from the North Carolina and Georgia lines. In the latter State several deposits are found, which form the continuation of the Ducktown ore-belt.

A heavily wooded, hilly tableland, some 1800 feet above the sea, surrounded by high mountains, met the view of the first explorers of the Ducktown region. Huge masses of iron ore (gossan) attracted the attention of Dr. Troost, as early as 1849. Even before this date, in 1843, a Mr. Semmons had discovered copper ores, but failing to recognize them as such, rejected them as refuse, and collected a quantity of iron pyrites which he mistook for gold. Some iron forges, established in the vicinity, using the iron ore of the gossan, had found their iron to be "red short" from the presence of copper.

The first shipments of ore from the district were made by a German mining engineer, a Mr. Weber, who, in 1847, sent two lots of ore, containing respectively 32 and 14 per cent. of copper, to the Revere Copper Company (of Boston). Systematic search for copper began in 1851, and soon great quantities of rich "black" ore rewarded the pick of the prospecter. Openings still exist where thousands of tons of ores, averaging from 30 per cent. to 60 per cent., were found and dug out at a cost but little more than the mere expense of shovelling.

The surface of the Ducktown district is cut up into quarter sections and fractions thereof. Almost every quarter was, at one time, the basis of a mining venture, and more than a dozen companies operated simultaneously in the hills.

According to a report by Drs. A. Trippel and H. Credner, the mines were opened in the following order:

Hiwassee,	August, 1850.
Cocheco,	October, 1850.
Tennessee,	October, 1851.
Polk County,	November, 1852.
Cherokee,	December, 1852.
Eureka,	April, 1853.
East Tennessee,	June, 1853.

Isabella,	July, 1853.
London,	September, 1853.
Mary,	September, 1853.
Callaway,	November, 1853.
Culchote,	February, 1854.
United States,	August, 1854.
Biggs,	August, 1854.

In the month of September, 1853, the production of seven of the mines reached 808 tons of 28 per cent. ore, and the aggregate shipments of eight of the properties, since their commencement, was 14,291 tons, worth over a million of dollars.

Many of the shipments of ore were made to England, as this was the only manner of realizing on the product. But the danger from the black oxides giving rise to spontaneous combustion, whenever moistened by water, and the necessity of utilizing low grade ore, soon led to the erection of works for smelting the ore on the spot. In 1854, two blast furnaces were erected at the Tennessee mine, but were unsuccessful. In June, 1855, Welsh furnaces were sucessfully started at the Eureka, converting the black oxides into a 54 per cent. regulus, which was shipped.

These black oxides are, in reality, not a pure oxide, though so called in miners' phraseology. They derive their color largely from the presence of copper glance, and are an intimate mixture of glance, oxide, carbonate, and sometimes finely divided native copper. Two analyses, by Dr. A. Trippel, show their constituents :

Oxide of copper,	5.75	3.80
Sesquioxide of iron,	1.50	.63
Sulphur,	18.75	25.40
Copper,	71.91	41.00
Iron,	.93	26.56
Soluble sulphates of copper and iron,	.72	1.78
	99.56	99.17

A pile of such ore, laid on a bed of cordwood and moistened, often ignites the wood below, and thus roasts itself without firing.

In the ruins of the various old smelters, scattered over the Ducktown district, a veritable history of the progress in copper-smelting is written. As already mentioned, Welsh reverberatory furnaces and small blast furnaces were the first metallurgical plants. The latter had a section about 12 or 15 inches wide by 18 inches deep, with one tuyere at the back. A number of these small blast furnaces were built into one huge block of masonry. They were charged from in front, and were not over five feet high. Mexican

adobe furnaces have about the same dimensions, but are superior in being accessible from all sides.

Improvements were soon made in the building of these furnaces. Separate brick stacks, with replaceable lining of soapstone rock found in the vicinity, became the rule. Two tuyeres, through the back wall, were also introduced. Whenever the lining was cut through by the corroding action of the slags, the furnace was blown out and relined.

About the time of the consolidation of many of the companies into one strong concern, the manager, Mr. A. Raht, led a new departure by building the furnaces with thin walls, strongly bound by railroad iron. A single complete central smelting plant was also established to treat the yellow sulphurets, which now alone remained. The rich black ores had been practically exhausted. With but slight modifications, this plant was in use until the district was finally abandoned several years ago, by reason of the largely increased cost of fuel and the low price of copper. At that time charcoal was hauled an average of over ten miles, and the supplies of wood were drawn from even longer distances.

The furnaces of the Ore Knob works, until recently in operation, are a faithful copy of the originals erected at Ducktown, and have been described in detail by Dr. Egleston, in the *Transactions* (Am. Inst. Min. Eng.).

The annexed map is a plan of the Ducktown smelting works, usually called the Isabella smelter from its location on the territory of that mine. (See Fig. 6, Plate II.)

The ores from the mines are brought by rail or wagon to the roast-piles occupying the middle of the valley of Potato Creek. The sides of the valley are flanked by the smelting furnaces—all clearly shown on the map.

Before proceeding to the description of the metallurgical treatment of the ores at Ducktown, a somewhat closer examination of the mines will be convenient.

Ducktown mines can be divided into three parallel ore zones, about one mile apart. Of these, the western zone comprises the East Tennessee mine, the London, Cocheco, and Hiwassee. As has been mentioned, the ore of this zone is slate-ore, and consequently silicious. The eastern range of the Polk County, Mary, and Callaway mines contains pyrhotite and yellow copper ore. Comparatively very little silicious ore is found, and that only at

or near the edges of the deposits. The mass of the ore is decidedly basic, and sometimes contains considerable calcite. The accompanying plan of the Mary mine, from surveys made by the writer, gives an idea of the huge size of these deposits. In the crosscuts, which extend some 70 feet, both walls have not been reached, nor has the end of the deposit been reached by the underground workings. The section deduced from the diamond-drill holes will also be instructive. (See Fig. 7, Plate III.)

The middle or Isabella range of deposits exceeds in size even the Western. A veritable mountain of gossan, covers an ore body penetrated at right angles with its strike for nearly four hundred feet, by a diamond drill-hole, without proving the foot-wall.

Judged solely by their contents in copper the Eastern Range stands preëminent, the Western next, and the Isabella last.

In 1877, when the bulk of the notes for this paper were collected, mining was prosecuted only at the East Tennessee and Mary mines. Both were connected with the smelter by a 3-foot gauge railway. During the summer of 1876, the ore shipped from the East Tennessee was mined from the edges of the great mass which was at that time being rapidly exhausted.

An average of 8 months' shipments of the sorted ore analyzed

5.5	per cent.	Copper
20.0	"	Iron
42.0	"	Silica
30.0	"	Sulphur

The stopes that had been formerly worked in the then exhausted centre of the bed, had averaged considerably richer, reaching 7.5 per cent. of copper. In the fall of 1876, and in 1877, the last of the bed on which the mine was originally opened was removed, and only occasional rich pillars and scraps of ore remained. These were shipped to the smelter to serve as flux for the basic Mary ores; an average of sixteen lots assayed

3.0	per cent.	Copper
74.0	"	Silica

It was about that time that the edge of a new mass was cut, but, like the last of the old bed, it averaged low in copper. Probably upon sinking and thus approaching closer to the centre of the new find, richer ore will be discovered.

The usual system of mining pursued at Ducktown was to sink a vertical main-shaft, crosscut to the ore, drift on this, sink winzes

connecting with the levels below, and break the ore in the stopes thus opened, by underhand stoping. Wages ruled $1.50 for miners, $1.25 for laborers, in 1876 and 1877. Most of the work was done on contract at the following rates:

Sinking main shaft complete, per fathom,	$295.00
Drifts, per fathom,	65.00
Winzes, per fathom,	145.00
Drilling (underhand), per lineal foot,	27 cents.
Tramming, per car or ton,	5 to 7 cents.
Breaking ore, by ton,	$1.00

The broken ore was hoisted without assorting and dumped over screens onto a sorting floor. There it was sorted into three classes, dead rock, stamp rock, and smelting ore. The last was cracked to fist size by men paid 50 and 75 cents per car of 2 and 3 tons. It was loaded at a cost of 8 and 12 cents per car, and sent to the roast heaps.

The screened ore and stamp rock were sent to dressing works. Of these there were two supplied by rock from the East Tennessee mine. Both were operated by water-power.

The East Tennessee works were an old 7-stamp wooden-battery of about 400 pounds, operated by an overshot water-wheel. Concentration of the stamped rock was effected by a Cornish tye and several buddles. From 18 to 20 tons concentrations were produced per month, assaying 6.2 per cent. to 6.9 per cent. metallic copper, and 30 per cent. insoluble residue. The tye tailings analyzed 2.1 per cent.; buddle tailings, 1.2 per cent.

The unsatisfactory results of these works led to their abandonment in 1876. At the Isabella dressing works much better and cheaper work was accomplished. The plant consisted of a rock-breaker, discharging through a revolving screen with 1½-inch holes, on to a revolving picking-table. The screenings passed to coarse power-jigs, and the tailings from these, together with the rejected ore from the picking-table, were automatically fed into a 600-pound 10-stamp battery of modern build. The battery crushed the ore through 10 mesh screens and fed a spitz kasten, which, in turn, fed 3 sand-jigs running at 160 revolutions per minute, and a revolving buddle.

In the summer of 1875, stamp rock sent to this mill averaged 2.3 per cent. copper; the picked ore from the table, 7.3 per cent.; the coarse jigged ore, 6.9 per cent; the hutch from these jigs, 6.4 per cent. With 3 per cent. stamp-rock, the works have produced

as high as 110 net tons ore, averaging 10 to 12 per cent. per month.

The results of a few months' work on very low-grade rock may be interesting, and are tabulated below:

1876.....	March.	April.	May.	June.
Rock assay.....	2.1 per cent.	1.96	1.75	1.0 per cent.
Concentration.....	9.1 " "	9.3	9.2 p. ct.	6.4 " "
Tailings	0.67 " "	0.68	.....	.....
Product.....	45. tons.	50.3 t.	53.9 t.	48.3 t.
Cost.....	.....	.....	$222.39	$182.61

As before stated, the works were operated by water-power derived from an overshot wheel. All employés at the mill were paid by the unit per ton of ore produced. To check waste, a deduction was made if the tailings assayed over two-thirds of 1 per cent.

Considering the low percentage of the stamp-rock, the above results are certainly eminently satisfactory. The concentrated ores were usually accumulated during the month to allow them to partially drain, yet even after the lapse of three or four weeks they still retained from 8 to 10 per cent. of water. That amount did not, however, interfere with the roasting of the ores in the usual manner.

Mining at the Mary mine was conducted on a system similar to that pursued at the East Tennessee, but the nature of the ore-body rendered some important deviations from the usual practice necessary. A rich streak of ore was generally found running along the hanging wall, and this alone was mined. The hanging was well defined and usually of quartz. Between the quartz and slate country a gouge was occasionally found. No attempt was made to mine the whole width of the ore-body, as at the East Tennessee mine, for the vast mass of the ore lying on the foot-wall was uniform and low grade throughout, so that sorting to materially increase the percentage was an entire failure. As a consequence, it became necessary to abandon the major portion of the ore-bed and mine a comparatively narrow zone of ore. This zone was both richer in copper and silica, and it alone was mined and sent to the cobbing-floors.

The following average samples of this ore clearly show its composition.

	1.	2.	3.
Silica	6.0	12.0	19.3
Copper	5.0	3.5	5.83
Iron	36.0	46.9	38.7
Sulphur	35.0	29 9	31.6
Alumina	.....	3.0	1.7
Magnesia	.....	.....	0.86
Lime	6.0	3.0	1.21
Carbonic acid, etc	.....	.....	1.1

Sample 1, by the writer.
Sample 2, taken and analyzed by Torrey & Eaton.
Sample 3, by Dr. E. D. Peters.

Other average samples by the writer were as follows: Average of 15 months shaft smalls (that is, the ore broken fine underground, and which should give a very fair average of the whole mass), 5.4 per cent. varying from 4.4 per cent. to 6.9 per cent.

The cars loaded in the stopes of the Mary mine were run out on to the cage, hoisted, and then trammed outside of the mine, and tipped on sorting-floors; a counterpart of those of the East Tennessee mine. Here the ores were broken, and sorted or culled into three classes.

1st. Rich smelting ores, shipped direct to the smelter after being cracked to usual size.

2d. Mundic or low-grade ore, principally iron pyrites, thrown over the dump.

3d. Silicious ores, sent to the Mary dressing works.

These works are a counterpart of the Isabella, except in the absence of coarse jigs. The ore sent over the dressing-table was hand-picked by boys, as at the East Tennessee, but the waste from the table was divided into two classes: First, silicious ores that were dropped direct into the feed-hopper of the stamps; and, secondly, mundic ores, which could not be enriched by concentration. These latter were thrown over the dump. An average of 43 large lots of "coarse selected ore," aggregating some 10,000 tons, and shipped from the Mary mine to the smelter in 1875 and 1876, assayed 5.9 per cent. copper. Smelting ore selected on the picking-table averaged 6.1 per cent., and mundic tailings from the same, 4.3 per cent. Concentrations of silicious Mary ore assayed from 5.9 to 6.8 per cent. copper. The tailings assayed 0.4 per cent. copper.

Wages, by the day, at the Mary mine, were $1.00 and $1.25, but

the miners earned at least 25 cents a day in addition by working on contract. The usual prices paid were as follows:

Drifts,	$ 75 per fathom
Winzes,	125 per fathom
Small shafts,	175 per fathom
Trammers,	10 cents a car, underground
Trammers in the adit,	5 cents a car
Loading,	8 cents a ton
Culling,	8¾ cents a ton
Inch drilling for dry back holes,	3 cents
For cutting in holes,	2½ cents
For underhanded holes,	2 cents

The following table gives the complete cost in detail of operating the Mary mine for a number of months:

Month.	Tons of ore mined.	Tons of ore shipped	Dead work, or lineal feet of development work.	Cost of tramming, hoisting, landing, culling and racking.	Cost of ore mining or breaking.	Cost of dead work.	Management and general expenses.
May	1140	239	41.7	847.11	974.24	1031.40	172.05
June	1906	1310	38.4	802.94	1151.69	737.91	221.25
July	1560	1040	31.2	707.87	1090.75	430.71	537.44
August	1650	1100	62.9	747.71	838.70	985.09	258.10
September	909	606	85.1	645.22	617.74	1216.34	171.56
October	1012	709	82.8	637.27	628.26	1319.48	186.73
November	802	549	70.9	922.17	574.80	1059.38	207.02
December	1020	676	45.4	650.00	750.00	799.70	208.56
January	1562	1032	42.6	799.17	1330.43	623.86	371.83
Totals	11,561	7267	50.1 ft.	6759.46	7959.61	213.87	2334.54

The cost per ton of ore broken as per the table is as follows:

Ore breaking,	68.8 cents
Tramming, hoisting, and landing,	38.5
Culling and cracking,	20.0
General expenses,	20.2
Dead work,	71.0
	$2.18.5

The ore wasted or thrown over the dumps was 37 per cent. of the whole quantity mined, and the cost of the smelting ore was enhanced proportionally, and amounted to not quite $3.47 per ton at the mine.

Dead work to an unusual amount was carried on during the months recorded. It was the desire of the management to develop the mine more rapidly than the immediate requirements called for. With a fair amount of dead work charged against the ore, the cost of mining it would have been about $1.75 per ton.

Should the mine ever be worked from wall to wall with machine drills run by air compressed by the use of the power of the Ocoee River (less than a mile distant), a practically unlimited amount of ore could be mined for $1.25 per ton. Three thousand tons per month could be broken in the ore-body as now developed, and further development would rapidly augment this amount. In commenting on the future of these ore-beds, we shall return to this subject.

The East Tennessee and Mary mines were the only ones operated in 1876 and 1877 on a considerable scale, and they were under the immediate supervision of the management. Ore was obtained by purchase from many of the other mines of the district, but this mining was left entirely in the hands of "tributers," who confined their attentions to the search for and extraction of "black ores," not removed by the former workers. The contents in copper of these ores varied greatly, but remained usually within the limits of 6 to 12 per cent. They invariably carried from 6 to 10 per cent. of moisture and always were silicious, carrying from 60 to 80 per cent. of silica.

A supply of these ores was a welcome addition to the produce of the mines. They formed excellent flux and were treated together with roasted first matte in smelting for black copper.

Smelting at Ducktown comprised the following six operations:

1. Ore roasting.
2. First matte smelting.
3. Matte roasting.
4. Matte smelting for black copper.
5. Blister smelting.
6. Refining.

Ore Roasting.

Roasting of the coarse yellow ores was conducted in piles in the open air. A few small roast-sheds, reserved for the roasting of the rich black, and concentrated or dressed ores, were in use. Much the largest part of the roasting was carried on without cover.

The "bottom," on which a roast-pile was to be built, was made of clay, stamped solid with wooden mallets. On this bottom two layers of cord-wood were laid at right angles and the ores were piled thereon to a height of 6½ feet. The size of the piles and the consequent contents in tons, varied between wide limits, being adapted to the locality chosen. In general, piles of 500 or 600 tons were aimed at, and the piles were never wider than 20 feet, this distance being governed by the railway tracks laid between the roast-piles. The top of a pile was covered with fine ore and the sides were allowed to take their natural batter. Fire was applied at the corners and sides, about every 20 feet. Once fairly burning, little further attention was paid to the pile until the fire became extinct. Dressed and black ores were roasted in smaller piles, averaging 30 or 40 tons. To create sufficient draught for the uniform kindling of such piles of fine ore, a wooden box was stood upright in the middle of each such piles. This served as a chimney until the ore was thoroughly ignited.

When the fire of a burning pile had become extinct, which usually occurred after the lapse of 8 to 10 weeks in the large, and about 4 weeks in the small piles, the ore was allowed to remain intact for a week longer until cooled, and then the pile was turned into the second fire. This was simply a repetition of the first operation, but care was taken to bury what was formerly the exterior of the pile in the interior of the new one. All lumps or sows formed by the melting of the ore, were also broken to fist size and scattered through the new pile. These sows, caused by too

great heat during the roasting, were, of course, undesirable, as they prevented the thorough oxidation aimed at, and pains were taken to avoid them by smothering the roasting with fine ore if too rapid.

The second fire, inclusive of the cooling, occupied a further lapse of 8 or 10 weeks, so that the ore was exposed to wind and weather a total of not less than four months. Of the undesirable effects of this exposure, more anon.

Mary ore had its contents of sulphur reduced by two fires, from about 35 per cent. to about 10 per cent. Slate ores lost a larger proportion of their sulphur, and rarely contained above 8 per cent. before smelting.

Ore roasting as thus carried out, was a very economical process in point of labor and fuel. On an average one cord of wood was consumed for 40 net tons of ore for each fire. The cost of labor in the first fire was 5 cents per 1000 lbs. for both Mary and East Tennessee ores; for the second fire, 7 cents and 6 cents respectively were paid; and for fine ores the pay was 12 cents per M.

The exact cost per net ton of ore was as follows:

$\frac{1}{26}$ cord wood, @ $3.00,	$0.15
Labor, 1st fire,	.10
" 2d "	.14
Materials,	.03
Total per ton,	$0.42

The losses of copper in the above-described roasting have been very generally ignored in judging of its expense. At least, proper emphasis has never been laid on them.

Owing to an unexplained difference of several hundred thousand pounds between the fine copper produced at the Ducktown smelter during a period extending over several years, and the monthly fine copper statements arrived at by deducting one and one-quarter unit from the assay-value of the ores produced, the writer's attention was forcibly called to this subject. A careful series of experiments were instituted; the results were rather startling. Repeated analysis of ore weighed into a roast-pile, and analysis and weighing of this same ore when sent to the matte furnaces proved an almost incredible loss.

From the large number of experiments and analyses, I quote the following striking examples:

Pile No. 349.—Mary Ore.

Gross weight of ore.	Water.	Copper.	Fine Copper.
399,213	2.5 per cent.	5.0 per cent.	19,461
204.444	2.0 " "	5 8 " "	11,620
95,182	3.8 " "	5.0 " "	3,617.
8,663	3.0 " "	5.1 " "	428
34,165	6.0 " "	4.0 " "	1,284
741,667 lbs. raw ore containing 36,410 lbs. Cu.			

The pile after roasting weighed 741,716 pounds—assayed 3.31 per cent. copper—equivalent to 24,985 pounds fine copper. 11,125 pounds copper, or 31.4 per cent. of the contents of the pile, had been lost while roasting. 170 days were consumed in roasting the ore, and 69 days in removing it to the smelting furnaces. Hence, the ore lay exposed to the weather for 239 days, that is, eight months.

Pile No. 447.—Mary Ore.

Gross weight of ore.	Per cent. water.	Per cent. Copper.	Fine Copper.
172,882	3.0	4.7	7,881
1,532	5.5	6.3	91
198,800	2.0	4.5	8,767
32,178	4.0	5.3	1,637
26,865	5.5	4.6	1,167
32,245	3.0	6.2	1,939
464,505 gross lbs. ore contained 21,482 lbs. copper.			

Weight of the roasted ore was 495,566 pounds, assaying 2.85 per cent., or 14,152 pounds fine copper. During an exposure of 186 days, the ore had lost 34.3 per cent. of its copper.

All the experiments made on a total of nearly 3000 tons of ore proved, beyond possibility of doubt, an average loss of more than one unit of copper, or over 20 pounds of ingot per ton of ore. This great loss during the roasting readily accounted for the deficit in the copper production, if only 1¼ per cent. was deducted from the assay-value of the ores for losses by treatment. The actual loss by the smelting process, as practiced at Ducktown, approached two units. Further experiments were made to confirm the results obtained. Experiments in roasting in furnaces, proved that no copper escaped in the fumes. This, indeed, was anticipated, as the heat in roasting never could reach a point at which copper is vola-

tile. The only other possible loss is by the leaching of the roast-piles during the heavy rains frequent in the Ducktown hills; and to this cause the great losses were finally ascribed. In referring to experiments in the leaching of these ores later on, this subject will be discussed in detail. Suffice it here to say, that with a roasting in one fire only from 1 to 1½ units of copper become soluble in water. The results were further confirmed by copper found in large quantity in the clay "bottoms" of the roast-piles. After a shower of rain, the roast-yard would be covered with pools of green water highly charged with copper.

Smelting for First Matte.

This operation is carried out in the row of eight furnaces shown on the general plan of the smelting works on the west side of Potato Creek. Two furnaces, with their dust-chambers, are always built in connection with one stack. The dust collected assays 4 per cent., and is put through a pug mill with 5 per cent. of clay dried and re-charged. A complete plan of such a pair of furnaces is shown by the accompanying drawing (Fig. 8, Plate IV.). This drawing explains their construction in detail, and renders further description unnecessary. The bottom of the furnace is built up of "steep," a mixture of burnt clay and charcoal, slightly moistened and rammed into place by a grooved iron tamping bar, measuring two inches square on the face. The tap-hole is made by placing a log of wood in position and ramming the steep around it, then pulling out the wood. The fronts of the furnaces are closed by common red brick. Smelting is carried on with cold blast delivered into a 24-inch main, supplying all the furnaces, by two 4-feet stroke and 4-feet diameter blowing cylinders, placed upright on an iron frame, and set in motion through connecting rods, cranks, and a large gear-wheel, by a 3-feet pinion on the driving-shaft of a 14 x 30 horizontal engine.

Power is furnished by 3 42-inch by 20-foot double return flue boilers, burning 4 cords of wood per diem.

Three men attend each furnace, one acting as charger, one as head of the trio or furnace-man, and the third as helper and slag-roller. The charges for each furnace are made up by a separate gang of men, on one or the other side of the charge floor. Meanwhile the charger consumes the ore mixture lying on the opposite side. A charge is called 23,000 pounds of ore, and is made up of the various ores, proportioned according to their contents in silica, iron

Fig. 2.

and sulphur, as determined by analysis, the aim being to produce an 18 per cent. matte. Foul slag, that is, such as contains 1 per cent. of copper or over, is spread on top of each charge. Carting a charge from the roast-yard and dumping it on the charge floor is contracted for $2. Levelling it off and spreading it properly cost 90 cents. Furnishing steeps per furnace was paid $12, and packing a furnace $12.50; all the work is contract. A furnace smelts on an average 12 tons charge per day, or about 11 tons of ore, producing from 5000 to 7000 pounds matte. One basket of charcoal, holding 2 bushels (of 2700 cubic inches) forms one charge, and this carries a burden of seven to nine shovels of ore. Unless the furnace works irregular the charcoal is charged against the front and the ore against the back of the furnace, so as to preserve the necessary nose in front of the tuyeres. Should this break silicious ore is charged against the back wall, which soon renews the nose. Slag runs continually into three-legged slag pots. Any particles of matte settle to the bottom, and are thus recovered. Each furnace is furnished with eight or nine of these pots and one slag buggy for changing and removing the pots. When the slag is chilled in the pot it can readily be removed by simple tipping.

Any matte or foul slag is then broken off as a "knob," and the clean slag is carted off at an expense of 5 cents per M. Matte is tapped at least twice a day. In a young furnace, where the crucible has not yet had time to be enlarged by corrosion, tapping becomes necessary every three or four hours.

Near the tap-hole of the furnace and to one side of the fore-bay a semi-spherical hole is dug in the floor of the furnace-house, and filled with steep. In this steep a basin is cut to receive the matte when tapped. As the molten matte chills to a depth of three-fourths of an inch on the surface of this bowl, the crust is removed and pulled off and to one side with iron hooks. By these means the matte is obtained in thin cakes, which are subsequently cracked into pieces about four inches square by contract ($7 per month per furnace), and the matte is then ready for the next process. Of the grave objections to this system the chapter on Matte-roasting will treat.

A campaign of the ore-furnaces while smelting the very basic Mary ores was of but short duration, rarely exceeding four weeks. Usually after the first week of smelting the crucible had become greatly enlarged by reason of the cutting action of the basic slags, and iron sows then began to form and accumulate in the bottom.

The first formed were easily removed by cutting down the forebay and chiselling them by means of steel crowbars. But soon the iron increased, and it then became necessary to blow down the furnace, take out the red brick front, and break off and remove as much iron as possible. After three or four chisellings the bottom of the furnace was built up solid with iron to a level above the tap-hole, and then the furnace was abandoned, and the campaign ended. A narrow space, left between the lining and shell proper of the furnace, formed an easy means of detaching the sow and adhering lining from the bottom of the furnace. The sow was then hauled out of the furnace-house on rollers, and carried off on the railway, to be thrown down some convenient embankment. No attempt to save the copper in the sows was made.

The rates of wages for the smelting help were as follows: Furnace men, $1.25 per day; chargers and helpers, $1; all help participated in monthly premiums for the best smelting.

The consumption of fuel in ore smelting was very great, averaging 40 bushels (of 2700 inches) weighing 22 pounds per bushel to the ton of ore, or a ratio of 880 pounds fuel to 2000 pounds ore, about 1 to 2½. This excessive consumption of fuel was due to the small height of the furnaces, a necessary evil by reason of the large percentage of iron in the charge. Flaming tops were the rule, and a consequent large loss of fuel. In part, the large charcoal bills were caused by the system of purchase of charcoal from contractors, whose aim it was to get the best pay possible, and who did not hesitate to deliver very inferior coal mixed with brands, even when strictly watched.

Charcoal made by the colliers of the company from sound hard wood was of superior quality, and cheap in comparison. Some few of the colliers were very expert in their trade. They would burn pits, holding 100 to 120 cords and yielding 100 bushels of excellent coal from three cords of wood. Charcoal so made cost as follows:

Royalty three cords wood,	$ .75
Hauling to pit,	.51
Chopping, at 45 cents,	1.35
Coaling,	1.25
Leaves (for cover),	.15
Hearth,	.04
Per 100 bushels (small bushels, at four pecks),	$4 05

Ox teams were the common conveyance of the coal from the pits to the smelter. An ox-team and driver usually loaded 75 bushels, and received $1.25 per day. One trip per day was considered fair progress. It would undoubtedly have been economical to build kilns at some distance from the smelter, for not only will the kiln coal carry a heavier burden, but 100 bushels coal will require a consumption of only 2½ cords wood under such conditions, a clear saving of three-fourths cent per bushel. The character of the slags made in the ore furnaces is illustrated by the following analysis of large average samples:

	I.	II.	III.
Silica,	37.20	33.25	31.0
Oxide of iron,	47 70	54.00	50.1
Alumina,	8.15		5.0
Lime,	2.52		
Magnesia,	4 80		
Oxide of manganese,	trace		
Oxide of zinc,	2.24		
Sulphur,	.17		
Copper,	.28	0.78	0.72

Number 1 was the product of smelting East Tennessee ore exclusively in the years 1874 and 1875. Nos. 2 and 3 are slags such as were common after the basic Mary ore formed the main portion of the burden. If, in the latter case, silica had been added to the charge to obtain a more acid slag, less copper would have been lost, but the amount of dead flux and consequent expense for fuel would have more than balanced the smaller evil of a large loss of copper in the slags.

Of the eight ore furnaces six were kept continuously in operation. The remaining two were in the hands of the masons, either repairing or drying. At one time the writer attempted to increase the capacity of these furnaces by an increase in width and the addition of another tuyere in the back. To compel the stock to travel uniformly and with regularity down the shaft of the furnace a four-inch partition wall was run down its middle about four feet. Practically the upper portion of the furnace was divided into two shafts. But the increased capacity (15 tons per diem) of the furnace was offset by the unwieldy size of the sow produced at the end of each campaign, and it became evident that until the smelting could be done without this inconvenience an increased size would prove a doubtful advantage.

The description of the Vershire and other modern furnaces will explain the simple means adopted to accomplish this object.

The first matte produced contained on an average as follows:

		Average.
Copper,	16 to 22 per cent.	18.7 per cent.
Iron,	40 " 50 "	45.0 "
Sulphur,	20 " 25 "	23.0 "
Insoluble residue,	1 " 5 "	2 2 "

The total cost of operating continuously six ore-furnaces, smelting 2000 net tons of ore per month, was as follows, in 1876:

80,000 bushels of coal, at 6⅔ cents,	$5,333.33
120 cords of wood for power, at $3.00,	360.00
Materials,	400.00
6 sets of furnace-men, at $240.00,	1,440.00
2 engineers,	100.00
Cracking matte, at $5.50 per furnace,	33.00
Carting and making charges, at $2.90,	504.00
Carting slag, at 5 cents per M.,	150.00
Steep—making and packing,	73.00
Removing sows,	72.00
	$8,465.33
Or, per net ton of ore smelted,	$4.23

Matte Roasting.

In the chapter on ore-smelting, we have left the matte lying, cracked to suitable size, in the ore-furnace houses. From there it is removed by carts, paid 12½ cents per 1000 pounds, either to the matte-roasting sheds, situated on the east bank of Potato Creek, back of the matte-smelting furnaces, or to the "Crusher." It was the custom at Ducktown to roast about one-third of the matte produced in reverberatory furnaces after previous crushing by a pair of rolls.

The object sought to be gained was a control of the percentage of sulphur in the charge for the next or black-copper smelting. Matte roasted in piles varied between wide limits in its percentage of sulphur and more or less of the thoroughly roasted crushed matte was added as the occasion required.

Had the matte roasted in piles been uniformly low in sulphur, as was quite feasible, the expensive furnace-roasting could have been saved.

The customary roasting in piles was done under cover of sheds

on clay bottoms, a counterpart of those of the ore roast yard. Two rows of wood were laid at right angles on a space about 25 feet square, and on this bed 60 tons of the cracked matte was piled and set on fire. Five fires were required to reduce the sulphur from 25 per cent. to about 10 per cent.

The time occupied aggregated, on an average, from 60 to 70 days. In the last three fires charcoal braize was mixed with the matte.

The cost of monthly treating 600 tons of matte by this system was as follows:

Carting matte, at 12½ cents per M.,	$150.00
Turning into 5 fires, at 7 cents per M.,	450.00
140 cords of wood, at $3,	420.00
300 bushels of coal, at 6⅔ cents,	20.00
Materials, clay, etc.,	40.00
	$1080.00
Or, per ton of matte,	$1.80

Furnace roasting was much more expensive. Two furnaces were employed,—one a two-hearth reverberatory, roasting 6000 pounds of matte, in 36 hours, and burning 2½ cords of wood; another a rotary furnace, 4 feet diameter, and 36 feet long, and doing double that quantity at the same outlay for fuel and labor. But the rotary only reduced the sulphur sufficiently by passing the matte through twice, and, as a result, the cost of treatment of the double mechanical roasting equalled that of the hand reverberatory. The sulphur, remaining in the furnace-roasted matte, was very uniform, quite the reverse of the roast piles, and ran from 9.6 to 10.2 per cent. in the mechanical, and 7.8 to 8.0 in the reverberatory furnace.

The average cost was as follows, per charge of 6000 pounds:

Hauling, at 12½ cents per M.,	$ .75
Crushing and hauling to furnace, at 20 cents,	1.20
Labor, at 60 cents per M.,	3.60
2½ cords of wood, at $3,	7.50
Materials, etc.,	0.20
	$13.25
Or, per net ton of matte,	$4.41

The system of roasting in piles, as practiced at Ducktown, although apparently economical, is in reality very far from being so.

The long time consumed, not only entails a very large investment for stock, but the degree of roasting is quite irregular and imperfect, and the clay-bottoms are the source of great expense. These bottoms adhere to the roasting-matte, prolong and prevent thorough roasting, and are finally incorporated in the mass by turning it into each successive fire, until the clay forms a veritable part, and as much as 10 per cent. of the whole weight of the matte. Eventually, all this additional weight must be passed through the next process, at a proportionate additional expense.

The following experiments, made by roasting matte on clay bottoms, and on a bottom paved with rock, serve to illustrate:

Roasting on Clay Bottom.

Weight of raw matte.	Weight roasted matte.	Time roasting.	Per cent. Sulphur.
120,588	142,022	62 days.	12.0
122,874	133,724	69 "	10.7
123,824	152,378	67 "	11.7
143,884	165,827	69 "	8.2
197,138	202,274	84 "	8.5

Average time in roasting, in 5 fires, 70 days. Sulphur reduced to 10.2 per cent., but the weight of the matte increased 11 per cent.

Roasting on Paved Bottom.

Raw matte.	Roasted matte.	Time.	Per cent. Sulphur.
136,938	136,610	42 days.	9.1
142,444	148,404	46 "	.....
141,166	150,562	53 "	4.0
158,248	157,980	.....	8 0

Average time of roasting, 50 days. Sulphur reduced to 7 per cent., and weight increased only 6 per cent.

It may here be noted that the third pile on the last table, in which the sulphur was reduced to four per cent., was roasted in what was practically a roast-stall, three of its sides being surrounded by rock walls about four feet high.

The evidence of the objectionable character of clay or other dirt mixed with the matte proving so striking, further experiments were

made to still further free the matte of foreign substances. With this end in view, casting the matte in sand beds and crushing the pigs in a rock breaker was tried, with the following excellent results.

A pile of 73.944 lbs. roasted in only four fires, had its sulphur reduced to 5.6 per cent. A fifth fire reduced it to 4.8 per cent. The roasting required only 42 days and practically no increase in weight was noted. In the absence of proper roast-stalls, as used for example at the Vershire mines (to be described hereafter), a rock bottom and the use of crushed matte is decidedly the most economical system of roasting. Even where stalls are employed,

FIG. 9.

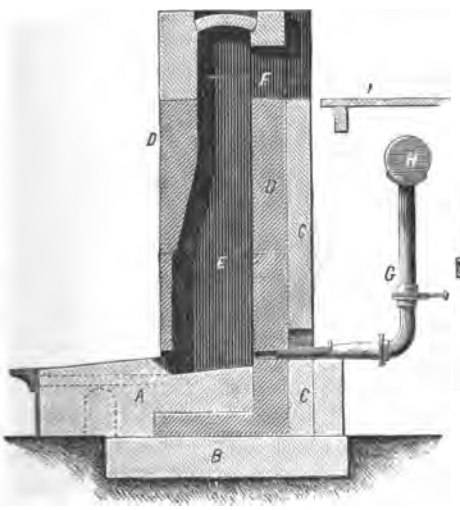
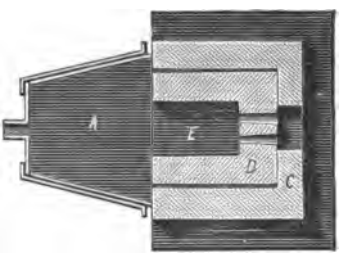


FIG. 10.



Matte Furnace.

In the drawings, B is the foundation of common rock; C the body of the furnace, also of rock; D the lining of soapstone; E the shaft of the furnace; A the steep packed into the bottom and forebay of the furnace; F the charging door; I charging floor; H blast main; G blast valves.

the casting of the matte in sand beds, or better still in iron moulds, and passing it through a rock breaker, is preferable to dragging the semi-fluid crusts of matte from off the matte basins and over the clay floors of the furnace houses.

Matte Smelting.

By reference to the general plan of the smelter, four matte furnaces will be found on the east side of the valley.

Their construction is identical with the ore furnaces, and they are blown by similar machinery, but of less capacity. The blowing cylinders have a diameter of only 3 feet by a stroke of 4 feet. Two boilers suffice for generating steam.

The smelting is very similar to the ore smelting. The roasted mattes are spread on the charge floors and are covered by roasted silicious black ores and foul slag from this operation, and any slag from the refining furnace. The black copper and second matte produced are cast in pigs in sand beds. If the charge contains as high

FIG. 11.

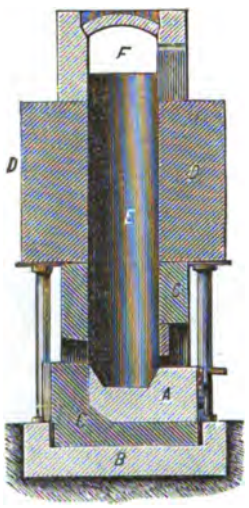


FIG. 12.

Water-jacket Furnace.

B, Rock foundation; C, soapstone lining surrounded by a water-jacket; A, steep basin; E, shaft of furnace; D, rock body of furnace supported on iron pillars and mantle-ring; F, flue to dust chamber and stack.

as 15 per cent. sulphur, very little black copper results. It is the aim to tap two-thirds of the product as black copper, assaying 90 to 92 per cent., and the balance as second matte, assaying 31.4 per cent. to 47.5 per cent., generally averaging over 40 per cent. The labor and consumption of fuel in matte and ore smelting are practically the same per ton of charge treated. Only the expense of repairs is greater in matte smelting. Four furnaces are necessary to keep two in operation. Two are required for repairs. A campaign seldom exceeds two weeks, and often after two or three days the furnaces break out and are badly cut. To remedy this trouble

the writer built, in 1876, what he believes to be the first water-jacket copper furnace tried in the U. S. A drawing is annexed. The furnace worked well. Its small size (2 ft. diameter) admitted of a capacity of only 10 tons per diem. This finally led to its abandonment. The proof of the perfect success of water-jacketing in preventing the corrosion of the walls, had however been made, for the section of the furnace when blown out was almost identical with its shape when first blown in.

Plans were at once prepared for a furnace 4 feet diameter, but the depression in the price of copper, which set in about that time, prevented the execution of the proposed improvement.

In smelting matte the furnaces rarely made sows. Indeed they were blown out the moment this accident seemed liable to occur. Should the black copper chill, it was broken loose at the end of the campaign and added to the next process. If the refuse from the repairs of the matte furnace contained an appreciable quantity of copper, it was added to the next charge.

The following is the detailed cost of smelting 600 tons first matte for black copper.

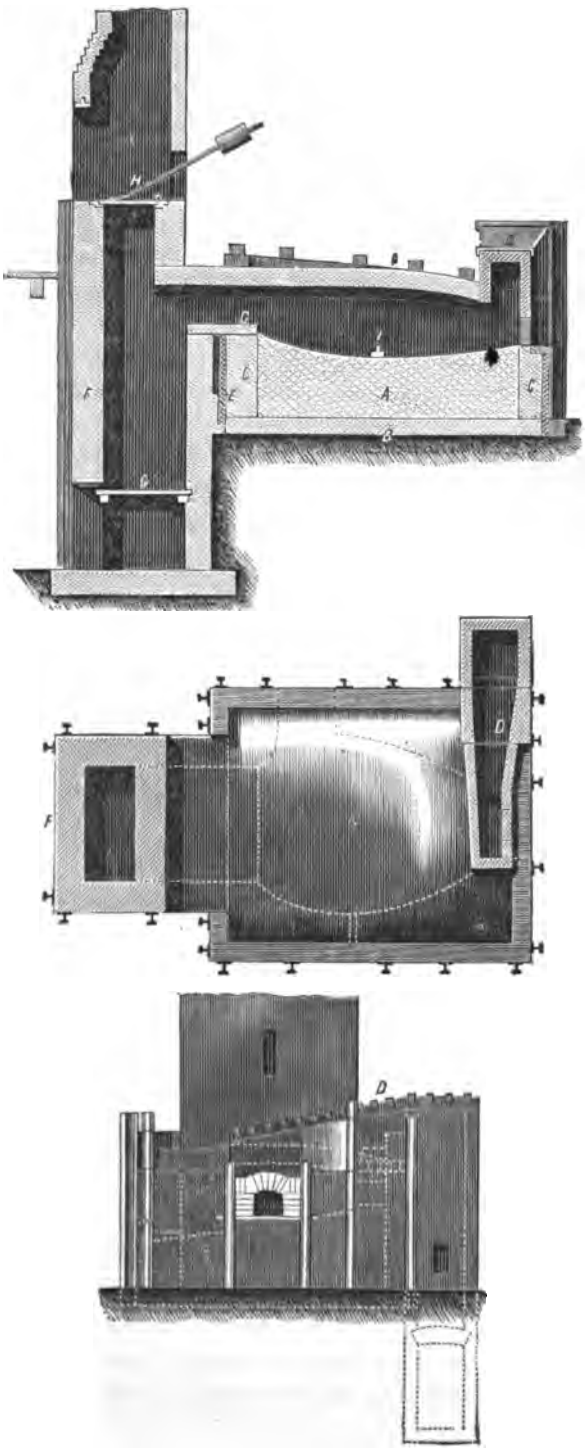
25,200 bushels coal, @ 6⅔,	$1680 00
105 cords firewood, @ $3.00,	315 00
Materials and repairs,	120 00
2 sets furnace men,	480 00
2 engineers,	100 00
Steep,	50 00
Relining,	50 00
Throwing back charges, @ 3 cts. per M.,	36 00
Carting slag, @ 5 cts. per M.,	50 00
Carting matte, @ 12½ cts. per M.,	120 00
	$3001 00
Or per ton of matte smelted,	$5 00

The slags made in matte smelting were very similar in composition to the ore furnace slags, as the following analyses of averages extending over several months indicate:

	1.	2.
Silica,	33.3	33.9
Oxide of iron,	64.8	54.5
Copper,	0.65	0.89

Any slag containing 1 per cent. of copper or over was classed as foul slag and was added to the charges of the matte furnaces.

FIGS. 13, 14, 15.



Ducktown Blister Furnace.

Pig Copper or Blister Smelting.

The black copper and second matte produced at the matte furnaces are carried a short distance to the adjoining Pig Copper furnaces. These are reverberatory gas furnaces connected by underground flues with a single stack back of the furnaces. Their dimensions and general construction are accurately shown by the accompanying drawing.

Wood fuel was used in the gas producer attached to the furnaces, which gave very excellent results. The furnaces proper were built of a shell of red brick, C, strongly bound by railroad iron. They were lined with the excellent "agalmatolite" mined about twenty miles from the mines. The fire-bridge roof and square flue leading to the main underground uptake, were built of the same material. The gas producer, F, was constructed of large blocks of fire-proof talcose rock found in the vicinity (the same that was used for the lining of the blast-furnaces). Only the top of the throat of the furnace was covered with fire-brick held as usual by iron cramps, D.

The bottom, A, was built up of crushed burnt quartz, semi-fused in place and seasoned with black copper scrap. I is the tap hole; G, the grate bars always covered with ashes; H, the lid of the gas producer.

Of the agalmatolite the following are analyses taken from Dana's *Mineralogy:*

SiO_2 Silica,	55 to 56
Al_2O_3 Alumina,	30 to 34
(K_2O) Potash,	5 to 7
(H_2O),	4 to 5

The rock was of the pure semi-translucent variety, readily cut and sawn to shape. The writer believes this rock to be superior to any brick ever made for copper-smelting. If laid with the grain towards the heat and carefully heated, it will crack but little, and once at a red heat the combined water of the agalmatolite is driven off, and it becomes almost indestructible and extremely hard, so hard, indeed, that the best steel will not touch it. Fire-brick laid alongside of blocks of the agalmatolite on the fire bridge of these furnaces was melted in four days, but the rock stood four months. A roof of this rock will wear away until but two inches of it are left, and remain perfect. With continuous work such a roof lasted nine to ten months, even though these furnaces were

treated with exceptional severity, by reason of the use of a strong air-blast (from the blowing engines), directed on to and under the matte and black copper as soon as partly molten. It was customary to charge the furnaces with 7500 or 8000 pounds of metal and matte towards the evening, and leave them in charge of a fireman overnight. In the morning the contents were red hot and the pigs of matte and black copper were melting on the edges. A four-inch blast-pipe, contracted to a two-inch nozzle, was then introduced at the side openings and luted in place with the nozzle pointing downwards and at right angles with the natural course of the gas flame. Rapid melting and oxidation immediately set in, the increase in heat due to the oxidation of sulphur by the oxygen of the blast being very marked. As soon as the charge was completely molten, the nose of the blast pipe was depressed until it touched the liquid metal.

In 1876 the writer purchased fire-brick tuyeres used in the bottom of Bessemer converters for this purpose, and forced these tuyeres into the molten matte as far as the limited pressure of the blast would permit. The use and effect of blast in desulphurizing matte had been a matter of common knowledge in Ducktown several years previous to that period. Soon after the complete melting of the charge, slag appeared on the surface and was drawn every ten or fifteen minutes through the front door.

This slag was a very fluid subsilicate, containing usually less than 30 per cent. silica and quite low in copper, sometimes not exceeding 2.5 per cent. A sample of the first skimmings analyzed 32.0 silica, 65.6 oxide of iron, and 1.1 per cent. copper. According as the slag was thought to be rich or poor in metal, it was sent to the ore or matte furnaces.

About five or six hours after the blast was turned on the charge was transformed into a very excellent quality of blister copper, and was tapped into sand beds. A new charge was then introduced through the side doors which were torn open. The bottoms of these furnaces were made as usual of crushed, burnt quartz rock, heated to incipient fusion and seasoned with black copper scraps. Blister smelters were paid $1.40 per 1000 pounds, and the total cost of treating 180 tons in two furnaces was as follows:

Labor of 360 M., @ $1.40,	$504 00
290 cords wood, @ $3.00,	870 00
Repairs and materials,	70 00
	$1444 00

Or per ton of blister not quite $8.00, or rather over one-half cent per pound of ingot.

Refining.

The operation of refining was carried out in a reverberatory furnace, built precisely like the blister furnaces, but of rather small capacity. Although it was possible to refine a charge of 10,000 pounds in the furnace, it was a rule to charge only 7000 pounds in the evening. During the night the charge was melted down and oxidized, so that at six in the morning samples showed the complete elimination of sulphur, by the absence of blow holes in the tests. The copper was then allowed to oxidize an additional half hour, and afterwards covered with charcoal and poled to tough pitch. By 8 A.M the charge became ready for ladling. The loss in weight from blister to finished ingot averaged 13 per cent. The refining slags were added to the matte furnace burden. Fifty-two hundred pounds of refined copper in ingots of fifteen pounds was ladled into the moulds by three or four helpers in about three hours' time. During this time the heat of the furnace was carefully regulated and tests taken at intervals. Should the copper lose its proper pitch, it was again oxidized and poled. The moulds used were made on the spot of refined copper, fitted by a groove into the usual iron shoes. Eight of these were strung on an iron bar fastened in a cast-iron box, filled with water to such a height that on tipping or revolving the moulds around the bar the surface of the hot ingot rested in the water, and cooling, contracted, which caused the ingots to drop into the water. Six or seven of these boxes, connected with a flowing stream of water, were in use. The cost of refining blister, if the refiner was kept continually employed, was as follows per month:

Labor and salary of refiner, ...	$500 00
140 cords of wood, @ $3.00, ...	420 00
450 bushels coal, @ 6⅔, ...	30 00
180 poles, @ 10 cents, ...	18 00
Materials and repairs, ...	60 00
	$1028 00

On a product of 150,000 pounds per month this cost is about 0.7 cent per cent. If worked to its full capacity of a charge of 10,000 pounds per day, the refining need cost but a trifle over one-half cent.

The following is the aggregate cost of smelting 2000 tons (net) ore per month at Ducktown in 1876.

Railroad transportation from mines to smelter—labor, $350; fuel, etc., $1.00,	$450 00
General expenses,	600 00
Roasting ore, 2000 tons, @ 42 cents,	840 00
Smelting ore, 2000 tons, @ $4.18,	8,360 00
Roasting matte, 400 tons, @ 1.80,	720 00
Roasting matte, 200 tons, @ 4.41,	882 00
Smelting matte, 600 tons, @ $5.00,	3,000 00
Blister smelting,	1,434 00
Refining,	1,028 00
Total,	$17,314 00
Or per net ton of ore treated,	$8 66
Cost of mining Mary ore per ton,	3 47
Total cost per ton,	$12 13

From this cost per ton of ore, the expense of making ingot copper can be readily deduced.

In 1875 all the ore mined at Ducktown yielded less than 80 pounds copper per ton, and hence the cost per pound of ingot would be over 15 cents, a cost which would be prohibitory under present conditions.

As we continue the examination of the various pyrites deposits, the conclusion drawn for Ducktown will be found to obtain elsewhere. All of these mines, if worked for copper only, have ceased to be valuable, and are not likely in the future to again become profitable as a source of copper *alone*.

(To be continued.)

A RECENTLY DISCOVERED METEORIC IRON FROM INDEPENDENCE CO., ARKANSAS.

BY WM. EARL HIDDEN.

In late years the discovery of quite large masses of meteoric iron has been of frequent occurrence, and it has become almost unnecessary to call public attention to them because of their great similarity. Unless of rare form or of unusual composition, lengthy

descriptions of them tend to a repetition of much that has been written before.

Where these masses are seen to fall, by competent observers, all particulars concerning the time, velocity, direction, and distance travelled are of value, and merit immediate and faithful record; especially as the data for these several particulars are very meagre and not wholly satisfactory.

The mass of meteoric iron hereinafter described and illustrated, was not seen to fall but was discovered in the surface soil, and thus its history is incomplete. Of the one hundred and thirty or more, known masses of meteoric iron, only about six were seen to fall, all the others being accidentally discovered in a manner similar to the one here noticed; while the exact date and hour of fall of almost all the stony meteorites is known, or closely approximated to. These wanderers in space make no choice of locality when they come down to us. They may be discovered in one place as well as in another. We know of no law prescribing the latitudes wherein they must arbitrarily fall. Attention is called to this because the writer has lately seen a map of the world, on which were indicated the places where meteoric bodies had fallen, and it was plainly evident that the great majority of these discoveries were in regions most densely populated. For instance, on the continent of North America, the region between the twentieth and forty-fourth parallels east of the Mississippi River, monopolizes the great majority of these occurrences; in Europe the same statement would apply to a region west of the Urals, between the forty-fourth and sixtieth parallels, and in India, between the tenth and thirtieth; while the great domains of Siberia, Africa, Australia, South America, British America, Western United States, and Alaska, present, according to the records, only a few scattering discoveries of these celestial visitors. All this shows to us that these very prolific regions, comparatively speaking, are only indicators of the immense number of these bodies which have fallen to the earth and which must be ultimately discovered in these as yet almost unknown areas as they become peopled. China, with her dense population and immense area, has kept within her borders all specimens and all data relating to her meteorites, and reasoning from analogy a very goodly number must exist there, discovered and undiscovered.

Altogether only about four hundred distinct "finds" and "falls"

of meteorites (stony and metallic) are recorded, which number is certainly not very great when we consider that it covers all historic time.

By referring to the larger engraving, Fig. 1, the reader will get a fair idea of the exterior appearance and shape of this new Arkansas iron. Its surface is pitted with ovoid depressions of various sizes, which lie with their longer axes in nearly the same general direction; this direction being parallel to one set of the lines of structure as shown on the smoothed and etched surface. The exterior was almost black in color, and looked blistered. No rusty appearance or alteration from oxidation was noticed on any part, which would seem to prove that this meteoric iron had not been long on the earth.

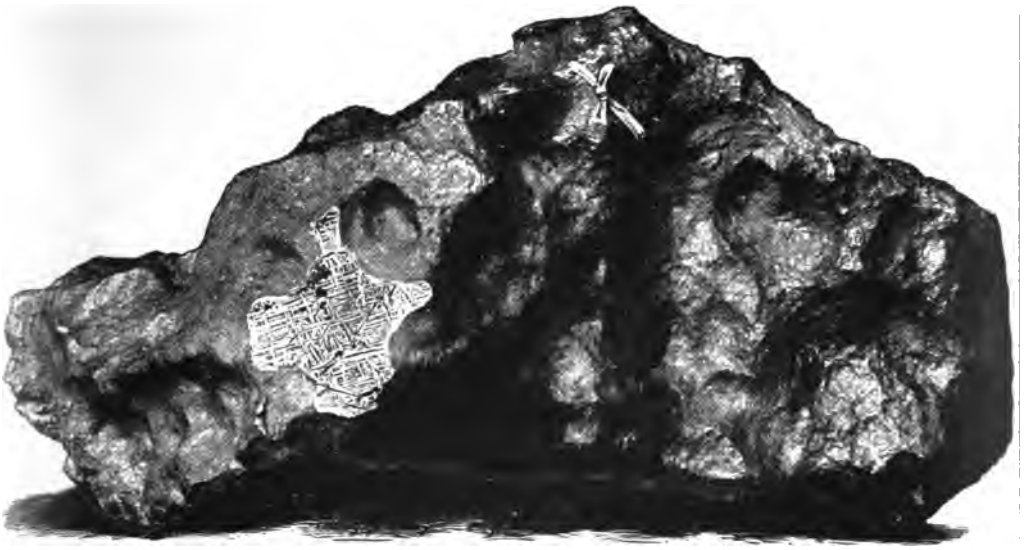
It was found in the latter part of June, 1884, in the manner set forth in the following extracts from a letter from Mr. John Hindman, of Elmo, Ark. He writes under date of July 2d, 1885.

"It was found about the last of June, 1884. My stepson, George Whitfield Price, accompanied by my son, John W. Hindman, and a young boy by the name of Monroe Marshall, concluded to take a ramble through the woods. They went along the north bank of White River to a mountain known as the 'Joe Wright Mountain,' a small eminence situated about seven miles east of Batesville, Independence Co., Arkansas. The soil there was cut into deep gullies, which, further down the mountain side, converged into one. It was where these gullies met that my stepson found the meteor. It had undoubtedly been imbedded a short depth below the surface, and as the earth was washed away, it became exposed and dropped to the bottom of the gully at the place where it was found. The town of Sulphur Rock is three miles S. W."

It was at the World's Industrial and Cotton Centennial Exposition that this mass of meteoric iron first came to the writer's personal notice. It formed a part of the very attractive mineral exhibit of the Arkansas section. It remained there until June, 1885, when it came into his possession. Its weight is ninety-four pounds. It is seventeen inches long and eight thick in its greatest diameter. Its size is unusual, yet several masses lately described excel it in this respect, notably, the large broken mass from New Mexico, and the "cigar-shaped" mass from Tennessee.

The most interesting feature is the presence of a hole through its edge, measuring five-eighths of an inch in diameter at its smallest

FIG. 1.



THE INDEPENDENCE COUNTY, ARKANSAS, METEORIC IRON.
WEIGHT NINETY-FOUR POUNDS.
(ONE-THIRD NATURAL SIZE.)

FIG. 2.

WIDMANSTÄTTEN LINES (NATURAL SIZE) ON THE INDEPENDENCE CO., ARKANSAS, METEORIC IRON.

part. The situation of this hole is shown in the engraving by a ribbon tied through it (Fig. 1). The length of the hole is one and three-quarter inches, being cone-shaped from both sides. This very remarkable feature is almost without a parallel among meteorites. It reminds us of that great natural ring of meteoric iron in the Smithsonian Institute which weighs more than half a ton, the aperture being large enough for a man to crawl through.

The presence of holes in masses of meteoric iron is difficult to explain. It would seem most natural for these bodies to come to us in the form of very symmetrical globes, instead of presenting most irregular and inconstant shapes. These holes may be caused by internal explosions, which, rending the sides asunder, leave, sometimes a bridge, or else a ring of metal.

The small surface which, in Fig. 1, shows faintly and in reduced size, the internal crystalline structure, is better illustrated in Fig. 2, which is a photographic reproduction of natural size.

Probably no better representation of the Widmanstätten lines—having a natural appearance—has been published heretofore from a type-metal plate. The use of the iron itself to print from after the manner of type is wrong, since the natural appearance is reversed. If a section was properly prepared for use after the manner of steel-plate engraving, results quite as good as this might be obtained.

The Widmanstätten lines in this Arkansas iron are remarkably perfect and abundant. Troillite (FeS), having a bronze color and lustre, was noticed as thin seams and veins on the polished face, and extending far into the mass.

An analysis by Mr. James B. Mackintosh, of a piece cut with difficulty from the edge, gave the following results:

Iron, ...	91.22
Phosphorus, ...	0.16
Nickel and cobalt, ...	8.62 by difference
	100.00

Nothing unusual thus appearing in its composition. It resembles closely the meteoric irons from Whitfield County, Georgia, and Chulafinnee, Alabama, described by the writer in the *Am. Jour. Science*, several years since.

I take pleasure in here expressing my thanks to Mr. Hindman for the information contained in his letter, and to Mr. Mackintosh for his kindness in furnishing the above analysis.

NEWARK, N. J., December 30th, 1885.

BOOK REVIEWS.

STORY OF THE HEAVENS. By R. S. Ball, LL D., Royal Astronomer of Ireland. With colored plates and numerous illustrations. London. Cassell & Co. pp. 551. $5.00.

Dr. Ball has here presented in popular form the latest results of scientific work in astronomy. The parts of the book likely to interest one quite familiar with the other books on popular astronomy, are the chapters on "Comets and Meteors" and "Tides." In the former chapter he explains in simple language the theory of Bredechin in regard to the formation of comet's tails.

In the latter chapter he presents his views of the relations between the moon and the earth in the past and in the future. He considers that in ages past the moon was close to the earth, and that she is gradually moving away; that millions of years ago, on account of the nearness of the moon, great tides, many hundred feet high, were raised on the earth, and reciprocally great tides were raised on the moon, then in a condition to be affected by tides. Our readers will recall that in a letter to *Nature*, some time ago, Dr. Newberry combated this view on the ground that there are no geological evidences of such great tides. Dr. Ball bases his views on the mathematical work of G. H. Darwin in an investigation of the motions of the satellites of Mars.

The opinions set forth in this chapter have met with considerable criticism, and are not accepted by all scientific men. The book is well worth reading, and is to be found in the College Library. J. K. R.

THE SQUARE BAR MICROMETER. By S. C. Chandler, Jr. JOHN WILSON & SON, Cambridge, Mass. 1885. Pamphlet, pp. 20.

In this monograph Mr. Chandler, of Harvard College Observatory, shows the methods employed in working with a square bar micrometer. The mathematical formulæ for the reduction of observations and the formulæ for the corrections are elegantly developed, and put in a convenient working shape. The paper shows the greater advantage of the square bar over the ordinarily used ring micrometer.

The pamphlet is mainly of interest to the working astronomer.

J. K. R.

PHOTOGRAPHIC MOSAICS FOR 1886. EDWARD L. WILSON: Philadelphia.

The widespread fraternity of amateur photographers is laid under an annual obligation to the editor of the *Philadelphia Photographer* for the many useful notes included in the Mosaics. The present number contains several contributions on methods of development, remarks on apparatus, a method of taking instantaneous views on slow plates, which in itself is worth more than the price of the book, and notes on many other subjects of interest. Receipts for cements, pastes, lacquers, etc., of more than photographic usefulness, also find a place. †

LIST OF TESTS (REAGENTS). *Arranged in alphabetical order according to the names of the originators.* By Hans M. Wilder. P. W. BEDFORD: New York. 1885.

This is a compilation of different special tests, which have been suggested by various authors from time to time, and which are generally referred to by the authors' names. As in many cases the reader may

have never heard of the test before, he is naturally somewhat at a loss to know what to do. With this list by his side he simply turns to the name, and in all probability will find a short description of the process used and the reaction to be expected. The value of the work would be much increased with but little extra trouble by incorporating in tabular form the reactions of the alkaloids, oils, resins, etc., so that they could be easily compared in employing the tests described. †

ABSTRACT.

Flood Rock Explosion.

General Abbott has sent to *Science* advanced sheets of his report on the "earthwave at the destruction of Flood Rock." This report shows that in the destruction of the rock 48,537 pounds of dynamite No. 1, and 240,399 pounds of rackarock were used. The photographs show that all parts of the mine were fired, practically, at the same time. The instant of firing was recorded to be 11 h. 13 m. 50.2 s., eastern standard time.

The table below gives the results of the observations made at several places as to the velocity of the earth-wave. At all the stations the observers watched a surface of mercury to which the reflection of some small, well-defined object could be seen. The arrival of the disturbance shook the mercury and caused the reflected images to disappear or to vibrate.

Station.	Distance in Miles.	Interval of Transmission.	Velocity in Miles per second.
Willett's Point	8.33	8.5	0.98
Pearsall's, L. I.	16.78	6.6	2.54
Bay Shore, L. I.	36.65	13.0	2.82
Patchogue, L. I.	48.52	15.4	3.15
Goat Island, R. I.	144.89	58.8	2.46
Harvard Observatory	182.68	219.8	0.83
West Point	42.34	13.6	3.11
		10.9	3.88
		10.9	3.88
Hamilton College	174.37	45.0	3.88
		45.2	3.86
Princeton College	48+	51.	0.94

The velocities are not accordant, but all observations show higher velocities than those deduced from observations on natural earthquakes. —[*Science, January 8.*] J. K. R.

Note.

At the December meeting of the American Metrological Society, by invitation Mr. Frank Rees ('75) exhibited and explained his new form of repeating rifle. This weapon is arranged so that it automatically loads and throws out the exploded cartridge without the necessity of lowering the gun from the shoulder. Twenty shots can be fired, if necessary, in twenty seconds.

ALUMNI ASSOCIATION NOTES.

QUARTERLY AND BOARD MEETINGS.—We give below extracts from the minutes of the Quarterly Meetings of the Association, and also from such meetings of the Board of Managers as have been held since the publication of the May Number.

MANAGERS' MEETING.—At a special meeting of the Board of Managers of the School of Mines Alumni Association, held May 23d, 1885, the Committee appointed at the last meeting of the Board to arrange for the purchase of a larger interest in the SCHOOL OF MINES QUARTERLY, reported that they had purchased from the Chemical and Engineering Societies of the School of Mines an additional one-third interest, one-sixth from each Society; and that hereafter the Board of Editors would be made up as follows:

Alumni Association, ...	4 editors.
Engineering Society, ...	1 editor.
Chemical Society, ...	1 "

On motion the report was accepted, the action of the Committee approved, and treasurer authorized to pay over to each Society the sum of one dollar for each one-sixth interest in the QUARTERLY.

FOURTH QUARTERLY MEETING.—The Fourth Quarterly Meeting of the School of Mines Alumni Association was held in Denver, at the St. James' Hotel, on the evening of July 2d, 1885; Dr. Iles, the first Vice President, in the Chair. Dr. Iles delivered a short but able address upon the value of technical education for any one expecting to make the West his home. The Railroad Committee reported through Mr. Fowler that most of the visiting members had availed themselves of the special rates obtained by the Committee and that the roads offering such rates had done everything in their power to accommodate the members, but it was advisable hereafter before applying to the railroad companies for reduced rates to make more definite arrangements in regard to numbers.

FIRST QUARTERLY MEETING FOR THE YEAR 1885–86.—Held in Hamilton Hall, Columbia College, Monday, November 16th, 8 P.M. The Treasurer reported a balance on hand of $289.32. The Secretary reported the resignation of the Assistant Secretary, Mr. H. W. Leavens, and stated that he had arranged with Mr. T. B. Stearns to act as Assistant Secretary until a successor to Mr. Leavens could be duly appointed. The Secretary also stated that he was in receipt of a letter informing him of the death of Mr. F. D. Browning, class of 1880, at Colorado Springs, August 25th, and in view of the well-known high character and professional standing of Mr. Browning asked that a Committee be appointed to draw up appropriate memorial resolutions for the action of

the Association. The Chair appointed P. deP. Ricketts, James L. Greenleaf and Alfred L. Beebe such Committee. The Railroad Committee appointed to arrange for transportation to the West, reported substantially as stated in the extracts from the Denver Meeting of the Association. On motion the report was accepted and the Committee discharged. The Association then proceeded to the discussion of the question of elective representation on the Board of Trustees of Columbia College. The matter was referred to a Committee of five, with instructions to collect data relating to the desirability of such representation by conference with the Alumni Asssociation of the School of Arts, the Trustees of the College and others, the Committee to report at a future meeting of the Association.

The Association then discussed the second question mentioned in the notice calling the meeting, namely, the method of conducting commencement exercises as at present existing; on motion the matter was tabled.

Mr. Butler then presented the following resolution:

Resolved, That hereafter all questions brought before this Association for discussion shall be first presented in writing to the Board of Managers for their approval, or notice of the question to be discussed shall be given at the meeting of the Association preceding the one at which the question for discussion would come up. (Carried.)

In accordance with the foregoing resolution, Mr. Butler gave notice that at the next meeting of the Association he would bring up a plan for the reconstruction of the Board of Managers.

The meeting then adjourned to partake of the collation provided by the efforts of the Committee.

MANAGERS' MEETING, held December 28th, 1885. Treasurer reported a balance in the treasury, after paying the amount due the QUARTERLY, $316.07.

Mr. Beebe presented his resignation as one of the editors of the QUARTERLY, stating that he found it impossible owing to the press of other business to attend to the work required. On motion his resignation was accepted, and Mr. James L. Greenleaf, C.E., elected as his successor.

A motion was presented relating to the payment of dues and providing that all members three years in arrears should be dropped from the rolls. The motion was seconded, and an amendment offered to table the same until the publication of the Constitution and List of Members of the Association. Amendment carried.

On motion Mr. T. B. Stearns was appointed Assistant Secretary of the Association, and the action of the Secretary in relation to the resignation of Mr. Leavens and the Western office of the Association confirmed.

MANAGERS' MEETING, held December 30th, 1885. There being no reports of Committees or other business, the Board proceeded to the election of officers.

The Chair appointed Messrs. A. J. Moses and A. L. Beebe a Committee to audit the accounts of the Treasurer for the year. On motion the meeting then adjourned.

SECOND QUARTERLY MEETING FOR THE YEAR 1885–86, held in Hamilton Hall, Columbia College, Wednesday, December 30th, 1885.

The Treasurer reported a balance in the treasury of $300.07.

The Secretary announced the manager and officers for the year 1886, as follows:

OFFICERS: *President*, F. A. Schermerhorn, '68; *1st Vice President*, M. W. Iles. '75; *2d Vice President*, F. R. Hutton, '76; *3d Vice President*, W. B. Devereux, '78; *Treasurer*, J. K. Rees, '75; *Secretary*, P. deP. Ricketts, '71; *Assistant Secretary*, T. B. Stearns, '81.

MANAGERS: John A. Church, '67; F. A. Schermerhorn, '68; J. C. F. Randolph, '69; Elwyn Waller, '70; P. deP. Ricketts, '71; P. T. Austen, '72; C. A. Colton, '73; F. B. F. Rhodes, '74; J. K. Rees, '75; F. R. Hutton, '76; W. E. Hildreth, '77; W. P. Butler, '78; N. L. Britton, '79; A. L. Beebe, '80; A. H. Van Sinderen, '81; A. J. Moses, '82; J. H. Banks, '83; J. F. Kemp, '84; Leon Marié, '85.

A special Committee appointed at the last meeting of the Association to draw up a memorial notice of Frederick D. Browning presented its report. On motion it was resolved that the Report of the Committee be adopted, and the Secretary be instructed to enter the same on the minutes and to forward a copy properly engrossed to the family of Mr. Browning.

Mr. Butler presented informally his ideas for the reorganization of the Association and changes in the system of electing managers and officers. After some discussion it was resolved that a Committee of five be appointed by the Chair to prepare a plan for such reorganization and changes and present the same at the next meeting of the Association for its action. The Chair appointed Messrs. Butler, Reed, Goldschmidt, Van Sinderen and Beebe as such Committee.

The Chair stated that the report of the "Committee of Ten" had been carefully considered by the Trustees and the Faculty of the School of Mines, that many of the suggestions made had been or would be carried out. He congratulated the Association and the Committee on the result of their labors.

Mr. Braschi presented a carefully prepared paper on the desirability of the study of Spanish and presented a number of letters from graduates relating to the same. On motion the subject was referred to the Board of Managers for consideration.

The meeting then adjourned.

LIST OF MEMBERS.—The Annual List of the Members of the Alumni Association, giving corrected addresses, will appear in the next number of the QUARTERLY. Graduates of the School will please bear in mind that only the addresses of those who pay their dues and show an active interest in the Association by becoming members of the same will appear in this list. All names or changes of address must be sent to the undersigned on or before March 1st, 1886.

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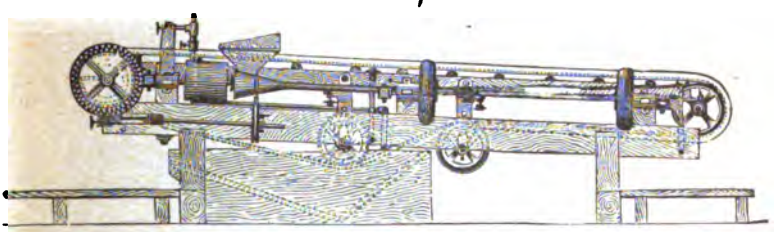

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MINING ENGINEER,

35 Broadway,

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SOME NEW MINING CASES.

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SINCE Dr. Raymond read his now celebrated monograph upon the "Law of the Apex" before the American Institute of Mining Engineers, more than two years have elapsed. In nearly all of our Western States, both Federal and Circuit Courts have been more or less actively engaged in the meritorious work of meting out justice and, incidentally, in construing our mining laws, but still there has been no attempt made, to my knowledge, except in short paragraphs in the columns of the *Engineering and Mining Journal*, to show to any except the members of the legal profession with what success, if any, this work has been accomplished. The Federal Supreme Court has contributed its share to the literature of the Mining Law, but not to the extent that was to be hoped for, for the reports contain but eight cases upon mining claims, and of these eight, one deals wholly with dissensions among owners, and one concerns a question which is solely one of equity. Nor has it contributed to a decision of the most important class of questions, for, as far as I can discover, the much vexed question of lateral rights is substantially in the same status in which Dr. Raymond left it, when he published his appendix to his "Law of the Apex," in February, 1884.

It is the object of this paper to review some of the cases decided during the past two years in the United States Supreme Court, and without entering into any elaborate discussion, to point out, incidentally, some of the more important points of controversy, which they are supposed to have determined.

The earliest case decided in this tribunal during the above mentioned period is that of Jackson *v.* Roby and another, decided December 3d, 1883, and found reported in 109 U. S. Reports, page 440. This was a suit begun in the United States Circuit Court for the District of Colorado, to determine adverse claims to mineral lands, under Section 2326 of the Revised Statutes. The act of 1872 of which this section is a part, declares that on each claim located subsequent to that date, there shall be annually expended for labor or improvements one hundred dollars, and on claims previously located, ten dollars for each one hundred feet in length along the vein, and further, in cases where the claims are held in common, the expenditure may be made on any one of them.

Section 2326 of the Revised Statutes is the one which defines the formal steps which a locator must take in order to secure a patent for mineral lands, and provides for the filing of adverse claims. In the case under consideration, Roby and his associates, the defendants, claimed the premises in controversy and made application for a patent. The plaintiff asserted his adverse right by reason of the same being a part of what was called the "Thomas Klak Claim," which he had purchased from grantors who had located in 1869, and showed upon trial that he was the owner of certain adjoining and contiguous claims known as "Lomax Gulch," which claims he began to work in 1872 and continued to work until 1880; and that in working his adjacent claim, he used a flume which extended a distance of one hundred and fifty feet over the defendants' claim, and discharged its tailings upon it, so that more than one-third of it was covered by this waste material. The plaintiff did no other work upon the claim, and attempted to trace his title from these acts, claiming that they were sufficient to give him a right of possession within the meaning of that clause of the section, which provides that where several mining claims are held in common, the labor or expenditure may be made upon one of them. The defendants showed that they had located a portion of the premises in July, 1880, and had purchased the remainder from prior locators, but failed to show that they or their grantors had done any work

at all upon the claim. The lower court directed the jury to find against both the parties, which it accordingly did, finding that neither party had proved title to the property. The effect of this verdict was to prevent the defendants from obtaining a patent, and to leave the plaintiff in precisely the same situation in which he was at the time of bringing suit.

The plaintiff appealed the case, and urged the clause above referred to relating to expenditure on one claim where the claims are held in common. The Supreme Court, however, following the case of Smelting Company *v.* Kemp, 106 U. S., p. 655, held that only such work or expenditure done on one claim could come within the meaning of the statute, when the work was done for or enured to the general benefit of all the claims, and that in the present case the work done upon the claim, *i.e.*, the depositing thereon of debris from Lomax Gulch, clearly was not for the benefit of the premises in controversy, and so the judgment of the lower court was affirmed.

The important point decided by this case is that the work done upon one claim, if it is to count under the statute, as to adjoining claims held in common, must be of equal beneficial value to all, and that where claims are held in common, if a general system is adopted, which will be beneficial equally to all of them, the expenditures required to be made in commencing said system or the work to be performed may be begun upon any one of them;—otherwise not.

In the same volume of reports, at page 550, is found the case of Sullivan *v.* Iron Silver Mining Company. This case was decided before Judge McCrary in Denver, on June 22d, 1883, and is reported in 16 Federal Reporter, 829. It was cited by Dr. Raymond in the above mentioned paper with the remark, "This is said to have been appealed to the Supreme Court. It is difficult to see how it can be set aside." It has, however, been set aside, but solely upon technicalities of practice.

In this case the meaning of the term "known lode," as it occurs in Section 2333 of the Revised Statutes, was construed. This section provides for the reservation of a known lode within a placer-survey, except in cases where the existence of a vein or lode within the placer-survey is *unknown*, in which case the patent for the placer claim carries everything with it.

In his opinion Judge McCrary held, that the term meant a claim

duly located or recorded and owned by a third party before the placer claimant applied to the government for a patent, and that the mere existence of a lode by geological inference, general rumor, or belief, did not take it out of the statute.

The grounds of reversal were as follows: The plaintiff as patentee of a placer claim brought an action to recover possession of a vein or lode within its boundaries. The defendants filed an answer alleging that the vein or lode was known to the patentee to exist at the time of applying for the patent and was not included in the application. The plaintiff demurred to it upon the ground that it did not set up any defence, that the defendants had located their lode claim within the ground covered by the placer claim, after the issuing of the placer patent, and that the plaintiffs, as owners of the placer claim, were not obliged, under the statute, to make application for a patent for the lode claim contained therein, unless discovered, located, and recorded by them at the time of applying for their placer patent; and Judge McCrary, in an exhaustive oral opinion, sustained the demurrer. The Supreme Court reversed the decision, on the ground that the answer well pleaded the facts which, under Section 2633, precluded the plaintiff from having any right of possession in the vein or lode, and was therefore not demurrable; but added, that the question of construction of the statute was not presented by the record for its adjudication, and that Judge McCrary had erred in going into it upon the issue of law raised by the demurrer.

The next case is Chambers *v.* Harrington, decided April 14th, 1884, and reported in 111 U. S. Reports, page 350. It is particularly interesting as being a reaffirmation of Jackson *v.* Roby, above considered. Like that case, it was a contest carried on under Section 2326 of the Revised Statutes, the facts being as follows: Harrington and other persons, the appellees, became the owners of three claims contiguous to each other—the Parley's Park, the Central, and the Lady of the Lake. They continued their work on the Parley's Park claim from 1872 until July 19th, 1878, when they transferred their operations to the Lady of the Lake, and did no more work on the other until September 13th, 1879, when one Cassidy, who claimed that the Parley's Park had been forfeited for want of work on it for more than a year, located a mining claim, which included a part of it. The Supreme Court of the Territory of Utah, to which the case was first appealed, found that

the owners of the three claims, "with a view to the future working and development of all three said claims, located what is called the main shaft in the Lady of the Lake surface ground. That said shaft is in such proximity to Parley's Park mining claim, that work in it has a tendency to develop said claim, and said shaft was located and intended for the purpose of developing all of said claims," also that work was of not less than $2000 in value. In view of the act of February 15th, 1875, amending Section 2324 of the Revised Statutes, so that work upon a tunnel, in a mine, should be held to dispense with work upon the surface, the Supreme Court affirmed the judgment, following the decision in Jackson *v.* Roby, above cited, and that of Judge Sawyer, in the case of Mount Diabolo M. & M. Company *v.* Callison, 5 Sawyer, 439.

Following this case, we come to that of Eilers *v.* Boatman, again an appeal from the Supreme Court of the Territory of Utah, found in the same volume of reports, a few pages further on (p. 356). This simply contains a reiteration of the practice, that under the act of April 7th, 1874, the Supreme Court must accept the findings of fact made by the Supreme Courts of the Territories as true, and, therefore, does not review the questions involved in the case, simply affirming the judgment. As this case, as appealed, is of little interest to the mining profession, I pass on to the case of Tucker *v.* Masser, 113 U. S., 203.

In 1881 the Supreme Court, in the case of Smelting Company *v.* Kemp, 104 U. S., 636, declared that a patent for a placer claim composed of distinct mining locations, some of which were made after 1870, and which together embraced over 160 acres, was valid. A similar question arose in the case under consideration. The plaintiff claimed ownership of three lots in Stevens' and Leiter's subdivision of the city of Leadville, in Colorado, by prior actual possession. The defendants denied the possession and averred ownership and right of possession. On trial before the Circuit Court of the United States in the District of Colorado, the plaintiffs offered proof to show occupation and erection of buildings upon the premises. The defendants introduced Stevens' and Leiter's patent of November 5th, 1878, which covered the premises and traced title, by mesne conveyances, to the patentees. The patent was for a placer mining claim, and the Court allowed the plaintiff to introduce, for the purpose of impeaching the patent, the record of proceedings in the Land Department upon which it was

granted. As it appeared from this record that four of the claims were located after 1870, and united into one claim, embracing about two hundred and ninety acres, the court held the patent void.

This decision has been reversed in the Supreme Court, the court stating that the same questions were fully considered in Smelting Company *v.* Kemp, and that it had seen no reason to question the soundness of the conclusions which it had there reached.

We come now to the case of Erhardt *v.* Boaro and others, decided on March 2d, 1885, and reported in 113 U. S., 527.

In the year 1880, Joel B. Erhardt, for many years United States Marshal for the Southern District of New York, seems to have employed one Thomas Carroll to search for mineral claims, on his and Carroll's joint account. This may be presumed with reasonable certainty from the fact that Erhardt laid claim to four-fifths of the location and Carroll only to one-fifth. The plaintiff's evidence tended to show that Carroll searched for valuable deposits of mineral on vacant, unoccupied land of the public domain in the Pioneer Mining District, in Dolores County, Colorado, and that by agreement between Carroll and Erhardt all lodes and veins discovered by Carroll, were located one-fifth in his name and four-fifths in the name of Erhardt. On June 17th, 1880, Carroll discovered the outcrop of a vein or lode of gold-bearing quartz, and forthwith located 1500 feet upon the vein designated the same as the Hawk Lode, and sunk a discovery shaft, about two feet deep. That upon the 30th day of the month, during the temporary absence of Carroll and Erhardt, Boaro and others came upon the ground, and, with knowledge of the plaintiffs' rights and claims, entered upon and took possession of the plaintiffs' excavation, removed the notice, and erected a stake and posted thereon a discovery notice of their own, calling the same the "Johnny Bull Lode." The evidence offered by the defendants in rebuttal tended to show, first, that when they entered upon the premises they had no notice of the plaintiffs' claim; second, that they commenced their discovery at a point some distance away from the plaintiffs' discovery shaft; third, that Carroll had abandoned his claim by reason of his failure to perform the necessary location work; and fourth, that Erhardt and Carroll had never claimed possession or asserted any title to the claim, until Boaro and his associates, by their working of it, had proved it to be valuable.

There was also direct evidence to show the date of defendants' entry, their compliance with the statute, and the amount of work performed by them.

The notice which Carroll posted at his discovery read as follows:

"HAWK LODE.

"We, the undersigned, claim 1500 feet on this mineral-bearing lode, vein, or deposit.—Dated June 17th, 1880.

JOEL B. ERHARDT, ⅘ths.
THOMAS CARROLL, ⅕th."

The defendants' notice, however, claimed 1500 feet "running six hundred feet northeast and nine hundred feet southwest and 150 feet on each side of the same with all its dips and spurs, angles and variations," and was dated June 30th, 1880.

The lower court, Judge Hallet, I am informed, assumed that the notice on Carroll's stake did not contain a specification or description of the ground claimed, within the meaning of the statute, and that the notice being deficient, any other discoverer could make a valid location upon the lode, at any other point beyond the actual one at which the discovery had been sunk.

This assumption the Supreme Court held to be erroneous, and therefore, reversed the judgment of the Circuit Court and ordered a new trial, and in pronouncing the opinion of the court, Mr. Justice Field says, "The written notice posted on the stake at the point of discovery of the lode or vein in controversy designated by the locators as 'Hawk Lode' declares that they claim fifteen hundred feet on the 'lode, vein or deposit.' It thus informs all persons subsequently seeking to excavate and open the lode or vein, that the locators claimed the whole extent along its course which the law permitted them to take. It is indeed indefinite, in not stating the number of feet claimed on each side of the discovery point and must, therefore, be limited to an equal number of feet on the course of the lode or vein in each direction from that point; to that extent as a notice of discovery and original location, it is sufficient."

The learned Judge further held, that, there being no local mining regulations in the vicinity of the Hawk Lode, the laws of the United States and of the State of Colorado were to be applied in the case. The statute of Colorado prescribes sixty days for making the excavations upon the veins from the date of discovery, and thirty

days thereafter for preparing the certificate and filing it for record, and further, that the discoverer, in addition to posting his notice, shall, before filing his certificate, sink a shaft upon the lode to the depth of ten feet from the lowest part of the same, under the surface, or so much deeper that a defined crevice shall be shown, and mark the surface boundaries of his claim. This work Carroll was prevented from doing by Boaro and his associates, and that, therefore, Carroll was not to be deprived of his legal rights by the wrongful act of the defendants, and that they being intruders and trespassers could not initiate any rights which would defeat those of prior *bona fide* discoverers.

A further matter of interest in this case, is the recognition which it contains of the present value of local mining regulations, as far as they are not modified by the legislation of Congress and the various States. For Justice Field says in his opinion, that "miners are still permitted, in their respective districts, to make rules and regulations not in conflict with the laws of the United States, or of the State or Territory in which the districts are situated, governing the location, manner of recording and amount of work necessary to hold possession of the claim." This recognition of Section 2324 of the Revised Statutes, coming from the quarter that it does, is very gratifying, and gives reason to hope that in the future the decisions of our highest court of appellate jurisdiction may be characterized by the same practical good sense, in questions which courts are so little apt to be competent to judge of as those which arise in mining causes.

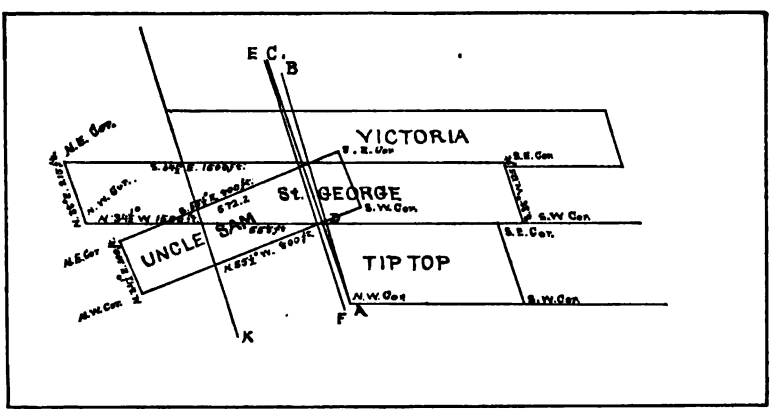
It may be thought by some that this decision goes very far. In my opinion, however, it is the most substantial evidence which the Supreme Court has given of its desire to uphold the rights of a *bona fide* locator in an action at common law, and to interpret the statutes, as far as such locators are concerned, in a wise and liberal spirit.

Immediately after this case, in the same volume of reports, is found the decision of the Supreme Court in an equity action between the same parties brought to restrain the defendants from committing waste on the premises while the action at law was pending. In this case the same court, speaking by the same Justice, has held, that an injunction will be issued to prevent irremediable mischief to an estate of this character, while the same is in litigation at law. Also a wise and commendable decision, and one

which will, I hope, put an end to much harassing and tedious litigation by deterring unprincipled persons from setting up adverse claims of the character of that urged by Boaro and his associates. This is the case which involves purely a question of equity, alluded to by me at the commencement of this article.

Another case which found its way into the Supreme Court, was that of the Richmond Mining Company *v.* Rose and others (St. George *v.* Uncle Sam), reported in 114 U. S., 576, brought there on writ of error to the Supreme Court of Nevada.

The contest was begun in the State District Court for the county of Eureka, by Rose and others, defendants in error, bringing a complaint, wherein, among other things, they alleged that since June, 1872, they had been the owners of the Uncle Sam claim on the western slope of Ruby Hill, and that the defendant, the plaintiff in error, the Richmond Mining Company, claimed an estate in fee in the premises, and had filed application for a patent therefor,



under the name of the St. George claim, and prayed judgment that the defendant might be barred from all estate or interest in the premises, or any part thereof, or any right of possession.

The answer of the defendant was filed in November, 1873, and denied any claim to the plaintiffs' location, except as it was covered by the St. George claim, which they admitted did cover a small part of it, but asserted a superior right as to such portion. The annexed sketch, taken from the report, shows the location of the various claims.

After a lapse of time of nearly eleven years the defendant

amended its answer, and set up that it had obtained a patent from the United States for all that portion of the ground in controversy, covered by the application for the patent for the St. George claim.

The lower court found that the plaintiffs had sunk their shaft on a mineral lode, and staked and marked out their claim, gave due notice of it, and did the necessary work on it to perfect the claim, prior to the operations of the defendant upon the St. George claim.

The judgment of the District Judge having been much modified in favor of Rose and his friends by the Supreme Court of the State, the Richmond Mining Company took the case into the Federal Supreme Court upon the following objections to the judgment of the Supreme Court of Nevada.

1. There being only three locators, each locator could appropriate, under both the Federal as well as State law, but two hundred lineal feet each, on the lode. The claim, being for eighteen hundred feet instead of six hundred, was, consequently, void.

2. That the court should have held, that the patent obtained by the Richmond Mining Company, for its St. George claim, should have prevailed (which covered everything which the defendant claimed), because it conveyed the legal title.

3. That the defendant, during the time while the litigation was pending, located another claim on the lode, parallel with the St. George, and outside of it, and called it the "Victoria," and procured a patent for it without notice to Rose and his friends, and obtained a patent for it without opposition. This patent the company set up on the trial of the case, claiming that, although it did not come in contact with the Uncle Sam claim on the surface, still, being upon the same vein, it gave the right to pursue the vein laterally under the Uncle Sam claim, and being a patent about which there was no controversy, must prevail in the suit.

These three objections are disposed of by the Supreme Court in the opinion by Mr. Justice Miller as follows:

As to the first objection it held, that when the statutes of the United States and local laws of a mining district, authorize a location on a vein of only two hundred feet, by each locator, a location by mistake for more than two hundred feet, is not thereby made entirely void, but is good for two hundred feet, and void only for the excess. It seems that the law allows each locator, who was the discoverer of the vein on which the location was made, an additional two hundred feet for his merit as discoverer. This fact

was recognized by the court in its opinion, but was not held to be controlling it in its decision in the case. Hence this point has been fairly settled upon the merits, without reference to subordinate rights. For Mr. Justice Miller in his opinion says: "We see no reason, in justice or in the nature of the transaction, why the excess may not be rejected, and the claim be held good for the remainder, unless it interfere with rights previously acquired." This is probably the most important point in mining law decided by the Supreme Court last year, and hereafter an excessive claim, if the excess is due to a mistake in location, and there is no outstanding older claim to the same land, is good and valid, to the extent of, at least, two hundred feet.

As to the second objection, the court held that the Richmond Company's St. George patent, so far as it affected the Uncle Sam, was void, and this for the following reasons: The Richmond Company filed its claim properly, and the plaintiffs made their protest within the sixty days fixed by law, viz., on September 24th, 1873, and on October 21st they began their suit by filing their complaint in the proper court in Nevada, but did not pay any docket fee or any other fee until August, 1874. The Richmond Company filed its amended answer on November 26th, 1874. There were negotiations from time to time, and the case was postponed from term to term, until March, 1874, when the last order postponing the suit, was made at request of the Richmond Company's counsel. In September, 1876, the Richmond Company produced before the register and receiver of the land office, the certificate of the clerk of the court, to the effect that the case had not been placed upon the trial calendar, nor any other proceeding been had thereon, from March, 1874, to the date of the certificate. The land officer proceeded to prepare the necessary papers, upon which the commissioner issued the patent. As far as the patent covered the St. George claim it was void, for the clerk's certificate was not sufficient evidence that the adverse claim had been waived. The State court had held that the delay was not the fault of Rose and his associates, and did not imply a waiver of their claim. Moreover, as the statute says, that all proceedings shall be stayed until the controversy is settled or decided by a court of competent jurisdiction, or the adverse claim waived, Judge Miller very properly interprets it to mean that they shall be stayed from the filing of the adverse claim, and not from the commencement of the suit in

court. Only the delay of thirty days, or the production of an instrument, duly signed and authenticated by the contestant, in open court, is made conclusive of a waiver of his claim. In the case in question, the suit having been begun, the court alone had jurisdiction, and until it had decided the issues the land office was powerless to act.

As to the third objection, it is disposed of by the opinion of the court to the effect, that the decision of the Nevada court "cannot be made nugatory by the introduction of a new claim by one of the parties, whose claim of right from the government, for this same location, is initiated while this litigation was going on," and that the subject-matter being before the court, the thing to be decided was the right of the conflicting claims to this lode, and that the parties were thrown back upon their original patents, and the result of the controversy was not to be defeated by the introduction of a new patent. The decision of the Supreme Court of Nevada was affirmed. An additional value of this case would seem to lie in the fact, that it declares what is in effect such a commencement of a controversy within the statute, as will stay proceedings in the land office, and that no settlement of the controversy short of an actual adjudication, or a declaration, in open court, of an abandonment of the claim, by one party or the other, will effect a dissolution of the stay, and confer jurisdiction on the land office.

The first point decided, namely, that a patent, if it by mistake includes an excess over the amount which any one locator can claim by law, is good to the extent of what the law allows, but void as to the excess, has been heretofore so decided by the lower courts, but has never, to my recollection, come up before the Supreme Court. This point can now be regarded as finally settled, and the doctrine well established.

The last case that comes within the province of this article, is that of Gwillim *v.* Donnellan (Cambrian *v.* Mendota), found reported in 115 U.S., 45. This is the most recently decided Supreme Court case, and has only recently appeared in permanent form. It was decided May 4th, 1885, and the opinion was written by Chief Justice Waite. It was a proceeding brought in the Circuit Court of the United States for the District of Colorado, under sections 2325 and 2326 of the Revised Statutes. The defendants

first applied for a patent, for the land covered by their location, and shortly afterward the plaintiff filed his claim.

It appeared from the evidence that Gwillim obtained title from one Thomas, who, upon the 16th of May, 1878, discovered in the public domain, a vein of rock in place, and located the premises under the name of the Cambrian Lode, and performed all the acts required by law. It also appeared that the defendants had obtained title to the Mendota through one Fallon, who had located his claim upon November 19th, 1878. It was admitted by the plaintiff, upon the trial, that after the location of plaintiffs' claim by Thomas, and before his conveyance to the plaintiff, a man by the name of Fallon had instituted proceedings to obtain a patent from the government, for another claim, including that part of Thomas' claim in which his discovery shaft was situated, and that no adverse claim was interposed, and Fallon entered his claim and obtained his patent, and that before Thomas did anything further on his claim, the defendants had entered therein and located the same upon the public domain.

The lower court instructed the jury to find for the defendants, for the reason that, "inasmuch as that part of the claim of said Thomas, wherein was situated his discovery shaft, had been patented to a third person, the plaintiff was not entitled to recover any part of the premises." This charge was affirmed by the Supreme Court. Hence it is apparent, that in suits to determine adverse claims, a plaintiff to entitle him to recover against a subsequent locator holding a patent, must show a location which entitles him to possession against the United States as well as against all other claimants. While Fallon was perfecting his claim, Thomas stood by, and set up no adverse claim. Thus Fallon obtained a superior title to the discovery shaft, and Thomas' location became invalid. Hence his right to apply for a patent became barred, and his location defeated, leaving all lands not included in Fallon's patent free to be rediscovered. In this respect the court follows Belk *v.* Meagher, 104 U. S., 284.

The case is chiefly of value in finally applying to mining cases the usual rule in ejectment cases (in the nature of which are cases brought under these sections), that the plaintiff must recover upon the strength of his own title, and not upon the weakness of his adversary's.

Herewith I have exhausted the Supreme Court docket. There

are many cases of interest in the lower courts, but as I said at the commencement, only Supreme Court cases are to be referred to in this article. To do adequate justice to the remaining class would, however, require more space than is at present at my disposal.

THE BISULPHIDE OF CARBON ENGINE.

BY W. P. TROWBRIDGE.

ONE of the most important and interesting facts brought out in the development of the science of Thermodynamics, was the law governing the theoretical efficiency of heat engines,—a law as invariable as any other law of nature, and capable of being enunciated in the most simple and direct language.

A very lucid and complete popular statement and explanation of this law can be found in the report of Rev. F. A. P. Barnard on the heat engines exhibited at the Paris Exposition of 1867. (Reports of the United States Commissioners, vol. iii.)

The simple fact that the theoretical efficiency of any heat engine depends upon the range of temperature through which the fluid acts; and that the maximum efficiency of any such engine, in which heat is applied at one absolute temperature T_1 and heat abstracted by the atmosphere or a condenser at a lower absolute temperature T_2, is represented by the ratio $\frac{T_1 - T_2}{T_1}$ whatever may be the nature of the fluid, is now so well known that it is hardly necessary to state it.

A popular demonstration of the principle is not, however, so simple, and, hence, reference has been made to Doctor Barnard's report, where the whole question is discussed in connection with ammoniacal gas-engines in a comprehensive but simple manner.

Inventors come forward, however, with almost periodical regularity at intervals of a few years, with propositions and devices for the saving of fuel in the development of power, by the use of some other medium than steam. The fluid, whose remarkable properties are to enable this to be accomplished, is, usually, ammonia, or the bisulphide of carbon. The idea which gains complete control, apparently, over all reasoning powers, not only on the part of the

inventors but, also, on the part of their confiding supporters, seems to be, that because these fluids exert a high elastic force at low temperatures and have low latent heats compared with steam, there must be, in spite of all scientific reasoning, a remarkable advantage in employing them in the cylinders of engines in the place of steam.

The glimmer of the wealth in store for those who shall be instrumental, through the use of one of these fluids, in "*saving one-half or two-thirds*" of all the fuel now burned in steam engines is the *ignis fatuus* that lures even sober-minded men, every now and then, into the mire of financial trouble.

It is generally useless to attempt to reason with inventors who insist that in the particular matter to which their invention relates, science is at fault; and even cool-headed business men often become infatuated with supposed discoveries, such as Keely motors, which are expected to revolutionize existing ideas in practical science.

The frequent attempts to introduce the bisulphide of carbon motor as a means of accomplishing a very great saving of fuel, notwithstanding the previous failures of similar efforts, and the steady and consistent warnings of scientific men, illustrate these facts.

The following discussions were prepared recently for a gentleman, who desired an investigation of the merits of such an engine, the operations of which in actual use disproved, it was said, all purely scientific reasoning on the subject.

It is needless to say that, although no practical tests were made of the performances of the engine, a somewhat prolonged use of it in actual work failed to show the promised diminution of the coal bills below those which would have been rendered for a steam engine working under similar conditions.

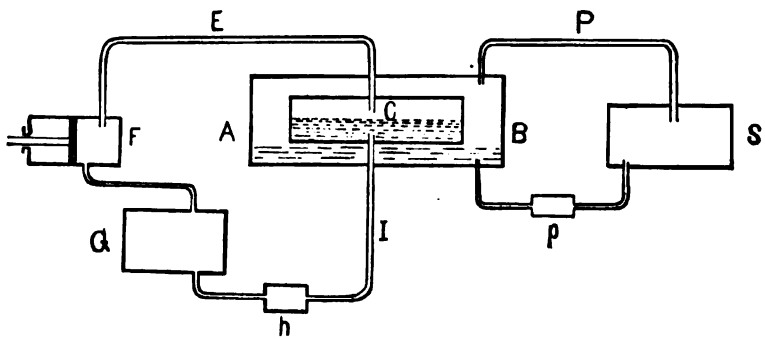
The first design made in this country for a bisulphide of carbon engine, was presented to the Novelty Iron Works, of this city, about twenty years ago. Previous to this, however, trials on a large scale were made in France.

The Novelty Iron Works' design was almost identical with those more recently proposed. The following sketch illustrates in a general way the parts or organs of the apparatus.

In the sketch, drawn in the most simple manner for illustration, S represents an ordinary steam boiler, which delivers steam through a pipe P to a large cylindrical chamber A B. Within this cham-

ber or shell is another, C, which contains liquid bisulphide of carbon. When steam is introduced into the larger chamber A B, the heat of the steam vaporizes a portion of the bisulphide, the vapor passing through a pipe E to an ordinary engine, where it acts like steam to give motion to the piston.

The exhaust from this engine is carried to a condenser G, from which the condensed liquid bisulphide is pumped by a pump, h, back to the bisulphide boiler through a pipe I. The condensed



steam in the shell A B, which comes from the liquefaction due to the heat imparted to the bisulphide of carbon, is pumped back to the steam boiler by a pump P.

The steam thus makes a circuit from the boiler S to the large shell and back without entering the working cylinder, while the vapor of the bisulphide makes another circuit, as shown in the sketch, through the cylinder, performing work and back to the interior bisulphide boiler C, without mingling with the steam. If no leakages occur, the same water, and the same bisulphide may thus work on indefinitely, each making its own circuit.

It will be seen that this apparatus may be changed into a condensing steam-engine by carrying the steam directly from the steam boiler S to the cylinder of the engine, thence to the condenser G, from which the condensed water may be pumped back to the boiler S,—by taking away, in fact, the intermediate shell A B and the interior bisulphide boiler C.

To determine the relative efficiencies of the two arrangements, *i.e.* of the bisulphide arrangement, and of the apparatus used simply as a steam engine, it will be sufficient to investigate the economical working of both under similar conditions, or by operating the

apparatus as an ordinary condensing engine, and then as a bisulphide engine.

In the actual apparatus the boiler pressure in S is reduced by a reducing valve before it enters the shell A B, while, at the same time, the cylinder F is steam-jacketed with steam direct from the boiler S. The bisulphide vapor then rises through the pipe E at a lower temperature than that of the steam in the boiler S, but at a high pressure; and is to some degree superheated in its passage to the working cylinder, inasmuch as the pipe E is also steam-jacketed.

To ascertain the efficiency of the bisulphide of carbon engine thus described, the following formula of Rankine is employed in what follows.

The work in foot-pounds performed by one pound of the vapor in expanding and performing work between the absolute temperatures T_1 and T_2 is given by the expression,

$$\overline{W} = a \text{ hyp. log.} \frac{T_1}{T_2} + b\,(T_1 - T_2) - \frac{c}{2}(T_1^2 - T_2^2) + V_2\,(p_2 + p_3)$$

in which the constants a, b, and c, are taken from the expression,

$$H = 107732.62 + 139.35.\ T - 0.2118\ T^2$$

or

$$H = a + b\,T - c\,T^2$$

which represents the latent heat of evaporation of one pound of bisulphide of carbon in foot-pounds.

T_1 is the initial absolute temperature.

T_2 is the absolute temperature.

Corresponding to p_2, V_2 the volume of one pound of vapor at the pressure p_2 (at the end of the expansion) and p_3, the back pressure.

The following table, giving the temperatures, latent heats, specific weights, and specific volumes of steam and bisulphide of carbon in British units, has been prepared from corresponding tables, giving the same quantities in French units, from Zenner's work on the Theory of Heat.

TABLE.—*Showing the temperatures, specific heats, heat of liquid, latent heats, specific volumes and specific weights of water* (H_2O) *and carbon sulphide* (CS_2).

Temperature.		Pressure in lbs. per sq. ".		Heat of liquid. From 32° F.		Latent heat of evaporation. Eng. units.		Specific volume. Vol. of 1 lb in cubic feet.		Specific weight. Wt. of 1 cubic foot in lbs.		Specific heat of liquid.		Specific heat of vapor.	
C.	F.	H_2O.	CS_2.	H_2O.	CS_2.	H_2O.	CS_2.	H_2O.	CS_2.	H_2O.	CS_2.	H_2O.	CS_2.	H_2O.	CS_2.
0	32	0.09	2.47	0.00	0.00	1091.70	162.00	2274.80	28.18	0.0004	0.0356	1.000	0.236	Volume of 1 lb of liquid carbon sulphide = 0.0128 cubic feet. Volume of 1 lb of liquid water = 0.016 cubic feet.	Spec. grav. of liquid CS_2 = 1.25 " water = 1.00
10	50	0.18	3.83	18.00	4.25	1079.19	160.31	1738.34	18.78	0.0060	0.0532				
20	68	0.34	5.76	38.01	8.53	1066.66	158.44	940.70	12.88	0.0010	0.0777		0.238		
30	86	0.61	8.41	54.02	12.83	1054.12	156.38	533.00	9.07	0 0018	0.1102		0.239		
40	104	1.06	11.94	72.09	17.17	1041.61	154.15	314.65	6.54	0.0031	0.1529		0.241		
50	122	1.78	16.58	90.15	21.53	1028.99	151.76	193.06	4.81	0.0052	0.2078		0.242		
60	140	2.87	22.52	108.25	25.94	1016.39	149.17	154.41	3.613	0.0064	0.2767		0.245		
70	158	4.59	30 02	126.36	30.35	1003.77	146.41	80.28	2.756	0.0120	0.3629		0.245		
80	176	6.84	39.31	144.51	34.81	1001.11	143.46	54.17	2 136	0.0190	0.4681		0.248		
90	194	10.14	50.65	162.69	39.27	978.42	140.35	37.34	1.679	0.0277	0.5956		0.249		
100	212	14.67	64.31	180.90	43.79	965.70	137.05	26.44	1.336	0.0378	0.7484	1.0132	0 251		
110	230	20.75	80.53	199.15	48.32	952.94	133.58	19.15	1.077	0.0522	0.9285		0.252		
120	248	28.78	99.58	217.46	52.90	940.13	129.92	14.05	0.876	0.0712	1.1415		0.254		
130	266	39.18	121.68	235.79	57.51	927.28	126.09	10.53	0.721	0.0950	1.3869		0.256		
140	284	52.45	147.16	254.19	62.14	914.37	122.09	7.99	0.599	0.1251	1.6669		0.257		
150	302	69.12	175.92	272.63	66.80	901.40	117.90	6.17	0.502	0.1620	1.9920	1.0262	0.259	0.480	0.965

This table serves to determine the quantities employed in the formulæ.

From p_1, the initial temperature, we get from the table $t_1°$, from which is easily obtained T_1.

If r be the ratio of expansion and v_1 the initial volume of pound of vapor, then we have $v_2 = rv_1$, and from v_2 we find from the table p_2 and T_2.

An engine recently constructed and kept in use for several months, was observed to be working under the following conditions:

$p_1 = 70 + 14.7 = 84.7$ lbs. per sq. inch $=$ 12196.8 lbs. per sq. foot.
$r = 4$.
$p_3 = 10$ lbs. per sq. inch $= 1440$ lbs. per sq. foot.
$v_1 = 1.034$ (from table) cubic feet.
$v_2 = 4 \times 1.034 = 4.136$ cubic feet.
$t_1 = 233\ 92$.
$T_1 = 693.32$.
$t_2 = 132.14°$.
$T_2 = 591.54°$.
$p_2 = 19.93$ lbs. per sq. inch $= 2869.92$ lbs. sq. foot.

$$\frac{T_1}{T_2} = 1.172.$$

$$T_1 - T_2 = 101.78.$$

$$T_1^2 - T_2^2 = 130673.05.$$

$$p_2 - p_3 = 1429.92.$$

$$\text{hyp. log. } \frac{T_1}{T_2} = 0.1588.$$

Putting these numerical values for the quantities which represent them in the expression for $\overline{W}$, we have

$$\overline{W} = 23340.6 \text{ foot-lbs.}$$

for the work performed by one pound of bisulphide of carbon under the conditions given.

The heat expended in foot-pounds to produce the same is given by the equation,

$$h = j\,c\,(T_2 - T_4) + a + b\,T_2° - c\,T_2^2 + \overline{W} - V_2\,(p_2 - p_3).$$

In which $J = 772$

$T_4 =$ absolute temperature of the condenser.

Giving the proper numerical values to these quantities from the tables, as before, we have

$$h = 139986.24 \text{ foot-lbs.}$$

The efficiency is found by dividing the work by the heat in foot-pounds required to produce it.

$$E = \frac{\overline{W}}{h} = \frac{23340.6}{139986.24} = 0.167.$$

For a condensing steam engine, under the same conditions, we may assume that the boiler pressure can be carried at 80 lbs. gauge or 95 lbs. absolute pressure, and that the pressure in the condenser is 4 lbs. We will then have for the steam engine, using the same apparatus, with the bisulphide attachments left off, and employing the same formula:

$p_1 = 13680$ lbs. per square foot.
$t_1 = 323.7°$.
$T_1 = 783.1$.
$V_1 = 4.5644$ cubic feet.
$V_2 = rv_1 = 18.2576$ cubic feet.
$t_2 = 231.3$.
$T_2 = 690.7$.
$p_2 = 21.29$ per square inch $= 30657.6$ per square foot.
$p_3 = 576$ lbs. per square foot.
$r = 4$.
$p_4 =$ pressure in condenser $= 2$ lbs.
$t_4 = 124°$.
$T_4 = 583.4$.

Substituting these numerical values, with the proper values of a, b, and c for steam in the formula for the work, $\overline{W}^1$, we have for the steam engine

$$\overline{W}_1 = 152888 \text{ foot-lbs. of work per lb. of steam.}$$

And for the heat in foot-pounds expended

$$h_1 = 971581.66.$$

From these expressions we get

$$E' = \frac{\overline{W}_1}{h_1} = \frac{152888}{971581.66} = 0.157$$

for the efficiency of the steam-engine.

These two determinations differ by only about one per cent., or one-hundredth part.

A remarkable coincidence when it is considered that each of these determinations depends on experimental determinations of the latent heats and the relations between the vapor-temperatures and pressures of each of them.

The result verifies, as was to have been expected, the law that theoretically the two engines are equal in efficiency.

There is one further consideration to be introduced, for which science gives us no precise practical solution at present. It is known that in the ordinary use of steam there is a waste of steam which does not appear in the ordinary theoretical formulas, due to the absorption of heat by the walls of the cylinder during the influx of steam into the cylinder and the cooling of the walls of the cylinder and the waste of heat during the efflux or exhaust.

This cause of loss or diminution of efficiency was first brought to the attention of engineers, I believe, by Professor Rankine in a paper read before the Institute of Engineers of Scotland, in 1852. In another paper, published in 1868, he discussed the confirmation of his previous deductions by Mr. Isherwood in the experiments made by the latter under the auspices of the U. S. Navy Department. A recent paper by Professor W. D. Marks, of the University of Pennsylvania, published in the Journal of the Franklin Institute, discusses the subject fully.

The loss from this cause in any particular case is uncertain in amount. The question, however, in the present case is not as to the actual amount of loss, but whether a loss from this cause, which must take place in the bisulphide engine, is greater or less than that for the steam engine. This is the only uncertainty in the comparison of the two engines. There seems to be no reason why one vapor should be more favorable than another in this respect, until it is demonstrated by experiments that steam and bisulphide vapor differ in this respect, we cannot assume superiority for either as far as cylinder condensation is concerned.

It is thus clearly demonstrable, not only by the application of the general laws of physics as determined by the science of Thermodynamics, but by special formulas applicable to heat engines in which the results of experiments on the physical properties of vapors are employed, that for the same initial and final temperatures there is no theoretical advantage in the use of bisulphide of carbon.

A similar discussion would show the same to be true of the vapor of ammonia.

THE PYRITES DEPOSITS OF THE ALLEGHANIES.

BY ARTHUR F. WENDT.

(Continued from page 188.)

IN 1875 the writer was tempted to investigate the question of leaching as applied to these ores. At that time the Hunt and Douglass leaching process had made a signal failure at Ore Knob, and that company had even then begun smelting operations. Both Ore Knob and Mary Mine ore contain an appreciable percentage of lime, and are generally similar ores. Especially is this similarity true of the low grade ores. Favorable results were, therefore, not expected from that process, but several tests were made of it on pile roasted ore. The results were extremely unfavorable. Instead of dissolving copper, the Hunt and Douglass liquor seemed rather to precipitate such copper as laboratory tests had shown to be soluble in water. An exposure of 4000 pounds of the ore for eight or nine hours to the Hunt and Douglass liquor usually resulted in an almost complete precipitation of the copper into a muddy sediment settling on the bottom of the tank.

The trouble was traced to the lime, as had been done at Ore Knob. So little of this sufficed to render the Hunt and Douglass process inoperative, that the mercantile value of that system of extraction cannot be said to be great. Only under exceptional circumstances, and with exceptional ores, can the system succeed. The laboratory experiments had pointed to water or weak acid as a better solvent of copper, and hence the following interesting experiments were carried out.

A round tank of about 150 gallons capacity was fitted with two wooden revolving blades, to keep the liquor and crushed ore in motion. A steam coil entered the liquor and permitted the employment of heat. A small experimental pile of low-grade ore was roasted in the usual manner in two fires, and allowed to weather for some weeks. Three months elapsed from the first roasting to the time of the experiments.

1. 3000 pounds of this ore, assaying 2.9 per cent., were digested for twelve hours with 150 cubic feet hot water. The water then held 45 pounds of copper in solution, or had extracted 1½ per cent. The residues from lixiviation assayed 2.1 per cent.

2. 4000 pounds of the same ore, assaying 3.3 per cent., were lixiviated for eight hours in 150 cubic feet hot water. Samples taken at that time proved the solution to hold 22 pounds copper. The stirring was continued for twelve hours more, and the liquor was allowed to settle. Thereupon it held only 10 pounds copper in solution, that is, more than one-half of the copper held in solution ten hours before had been reprecipitated by the lime. Residues assayed 3.5 per cent.

3. 4000 pounds ore assaying 3.3 per cent. were stirred with 150 cubic feet water acidified with one carboy sulphuric acid. The liquor tested 3 per cent. Beaumé.

After twelve hours lixiviation 65 pounds, or a little over 1½ per cent. copper, were in solution.

These experiments promised well, but the residues still contained a very considerable quantity of copper. This copper was almost entirely in the shape of sulphuret. It was noticed that on stopping the stirring blades, the ore containing sulphurets rapidly settled to the bottom and the oxidized leached ore remained in suspension. The action was that of the well-known Cornish "Keefe," a concentrating machine working on this very principle.

If the sulphuret could be concentrated so as to be again available as an ore, the question of its disposition was solved. With this object in view, the writer prepared a pile of low-grade ore. A good hard bottom was selected; on this only a single row of cord wood was laid, on a square of 28 feet. The ore was piled thereon and then covered to a depth of 10 inches on both top and sides with fine screened ore. Fire was applied. Cracks in the cover of fine ore were carefully closed as soon as they made their appearance, and the pile was roasted as slowly as possible, and then thoroughly

cooled before opening. As a result no agglomeration of the ore took place. Each piece of ore retained its original shape, and was barely adhering to its neighbors, and the whole ore was beautifully kernel roasted.

The kernel-roasted ore was crushed through a quarter mesh and then treated by lixiviation.

An average of the ore assayed 3.23 per cent. copper. The solvent was, as before, acidified hot water of 3° Beaumé. Analysis of the liquor was made every hour and sometimes every half-hour. Four thousand pounds ore were treated at a charge.

Time after charging.	Contents of Solution in lbs. of Copper.		
	Exp't 1.	Exp't 2.	Exp't 3.
½ hour.	.....	.....	52.0
1 "	.....	52.0	56.5
1½ "	.....	.....	52.0
2 "	.....	47.9	.....
2½ "	.....	.....	48.7
3 "	.....	47.0	.....
4 "	.....	43.8	.....
4½ "	.....	.....	39.0
5 "	.....	40.6	.....
6 "	33.3 lbs.	39.0	.....
7 "	30.8 "	37.3	.....
8 "	29.5 "	37.0	.....
9 "	27.1 "	} allowed to settle.	.....
10 "	25.8 "		.....
11 "	23.5 "		.....
12 "	22.7 "	36.4	.....

The experiments proved that within an hour after the beginning of the operation the maximum of copper was in solution. Subsequent stirring not only failed to dissolve copper, but the metal in solution was gradually precipitated by the lime in the ore.

The residues were composed almost entirely of the kernels and assayed 4.1 per cent., or nearly one per cent. more than the raw ore. They weighed about 1600 pounds per charge of 4000 pounds raw ore, and were in excellent shape to be again added to the roast piles.

Attempts at hand picking or separating the kernels from the inclosing crust were not successful. To select them with sufficient care to render the ore fit for smelting was too costly, and if rapidly and superficially selected they assayed only 4.2 per cent. Better

results were obtained by jigging. A partial concentration was effected. Sieve ore assayed 6.5 per cent., the hutch 5.6 per cent., and tailings 1.2 per cent.

The copper in experiment No. 3 would be accounted for as follows:

Cr.		Dr.	
4000 lbs. ore, @ 3.23 = 129.2 lbs.		Held in solution, . . .	56.5 lbs.
		1600 lbs. residues, @ 4.1 per cent., ...	70.0 lbs.
		Accounted for, . . .	126.5 lbs.
		Loss, ...	2.7 lbs.

It is manifest that very low-grade ores can be profitably treated by this process. The cost of the roasting would be very small, as only one fire, and half the usual bed of wood for this one fire, is needed. Twenty cents per ton will cover the whole expense.

Crushing the roasted ore at Ducktown with water-power will not cost to exceed 20 cents per ton. If one charge of 4000 pounds is worked every three hours in the lixiviating tank, or 16 tons per diem, the whole expenses will not aggregate $1.00 per ton, or about 2 cents per pound of copper.

Sulphuric acid of sufficient strength can be obtained by forcing gases from roasting pyrites into hot water, or better still into the lixiviating tank, and not even the expense of an acid-chamber need be incurred. The principal expense would be the scrap iron for precipitation. This would cost not less than 1⅛ cent per pound at the mines, or rather less than 1¼ cent per pound of copper, for the consumption of iron will be small owing to the presence of sulphurous acid. If the cost of mining these low-grade ores is taken at $1.25 per ton, and the total quantity of copper extracted at only 60 pounds per ton, the cost of refined ingot by this system should not exceed 6½ or 7 cents per pound.

Over the Rio Tinto process of treating the fines of copper pyrites, by gradual disintegration of the piles and slow extraction of the copper as a chloride, this system has the advantage of speed, certainty, and the saving in both salt and cost of plant.

Over the Agordo process of kernel roasting and hand-sorting of the kernels for further treatment by smelting, the process has the advantage of being a mechanical system and one adapted to the wages ruling in this country.

At Agordo the kernel pickers are paid about 18 cents per day.

They pick one ton of ore, selecting 200 to 225 pounds of kernels. Of course that method is impossible in America, with wages four to five times as high.

An application by the writer for a patent for the process described is pending before the U. S. Patent Office.

The analyses in the foregoing pages were made by the writer except where the analyst is especially named.

All samples were true averages and carefully drawn. They weighed as much as five pounds for every ton of ore or furnace product sampled. A skilled person was continuously employed in this department. The analyses were made by a modified system of volumetric determination of copper by cyanide of potassium, invented by the writer.

Experiments had proved the common method to give results generally from 0.2 to 0.3 per cent. too high, but always unreliable.

The modified course of analysis was as follows: 2 grams of the finely-powdered ore were digested in common muriatic acid, just sufficient to cover it in a covered casserole, until the evolution of gas ceased. About 5 c. c. strong nitric were then added. Violent and rapid disintegration and oxidation of the ore set in. The casseroles in number up to forty (or twenty samples in duplicate) were set on a large sand bath, kept at a low uniform heat, and allowed to evaporate to dryness. They were then removed to the laboratory desk and the dry mass moistened with conc. hydrochloric. Meanwhile 40 funnels had been prepared and set over the same number of beakers (about 3 inch). Each beaker contained a square piece of pure metallic zinc just covered with water. The contents of the casseroles were now washed with hot water into the funnels, and the residue on the filter paper washed repeatedly with hot water. This whole operation barely consumed an hour, and when finished the copper was found already precipitated by the metallic zinc in the beaker. If properly done it was in one spongy red mass and not oxidized. The zinc was removed by a pair of pincers and washed. The copper immediately settled to the bottom and was washed by decantation (three times usually sufficed) and finally dissolved in a few drops of nitric, heated to expel nitrous acid, and ammonia added and set aside to cool. Meanwhile the liquid from which the copper had been precipitated, together with the washings by decantation, had cooled and contained the iron of the analysis as a ferrous salt. This was at once

titrated with bichromate of potash, thus determining the iron. The ammonia-copper solution was then titrated with cyanide, determining the copper. Finally the filter papers, which had meanwhile been dried, led to the determination of the insoluble residue.

The three determinations gave every index required for the conduct of smelting operations. By this system the copper was determined in a state exactly like that of the pure copper used in standardizing. Even the amount of ammonia-copper liquor and the quantity of copper in the solution were the same. Under such conditions the cyanide determinations gave accurate results corresponding closely to battery determinations.

Georgia.

Turning from the Ducktown ore beds and following the strike of the rocks to the southwest, the first beds of pyrites are found in Fannin County, Georgia.

Some twenty years ago extensive operations were there carried on. A bed of ore resembling the East Tennessee mine or the Ducktown "Slate Ore" formed the basis of exploitation.

The ore bed was over 20 feet wide as shown by old workings now filled with water. Ruins of a concentrating plant and large heaps of silicious tailings give evidence of the character of the ore.

A ruined smelter, partially hid by the young forest growth, adds to the desolation of the scene now rapidly returning to its pristine wilderness. A number of other mines of no great magnitude were formerly worked in this vicinity, but even their location is now forgotten.

Pyrites from two Georgia mines has been used in the manufacture of sulphuric acid. One of the mines is that of the Tallapoosa Copper Reduction Company in Haralson County; the other mine that of the Paulding County Mining Company in Paulding County. The last named mine is on the East Tenn., Va. and Ga. Railway, 30 miles from Atlanta, where the ore was burned. The ore-bed is said to be only 5 or 8 feet in width. No large amount of ore has ever been shipped, and the small dimensions of the ore-body render it improbable that the mines ever become important.

Stone Hill, Ala.

The Stone Hill beds of pyrites in Cleburne Co., Ala., are the next noted occurrence of pyrites beds to the southward from Ducktown.

They are reported as very similar to the East Tennessee belt of Ducktown. Two deposits are known, but only one has been at all developed. It is about 1200 feet long and dips at an angle of 45° to the east. The strike is northeast, conforming to the general rule. A slate vein is distinguished about 15 feet wide, and beyond this mundic and pyrites are scattered through the mica schist for from 25 to 50 feet of additional width. In 1877 R. P. Rothwell examined this property and reports the average of the slate vein as assaying nearly 5½ per cent. of copper. Forty per cent. of this was selected for smelting and averaged 7.6 per cent. The balance was dressed in a 5-stamp mill, a counterpart of those described as in use in Ducktown.

The tailings from this mill assayed 1 per cent., and an average of many analyses of the dressed ore assayed as follows:

Copper,	6.81
Iron,	41.38
Sulphur,	30.56
Insoluble residue,	21.10

The metallurgical treatment consisted of burning the selected ore in small piles occupying a space of time of about five weeks, and of roasting the dressed ore in charges of 6000 pounds per 12 hours in two furnaces.

The mixture was smelted in reverberatories into a matte of 16 to 20 per cent.

This matte was then roasted in three fires in stalls, in about nine days' time, and then smelted in charges of 3000 pounds with 400 pounds slag from a previous charge to a second matte. The second matte assaying 30 to 40 per cent. was sacked and shipped.

A blast furnace was tried at these works but abandoned owing to the trouble with iron sows. Charcoal was cheap, 5 cents per bushel; wood, $1.25 per cord; hauling to railroad some 30 miles, $8.00 per ton. It is probable that large works could mine and smelt these ores for $10.00 per ton or produce copper for 10 cents per pound, until such time as the rise in the value of fuel would naturally increase the cost.

No great capital was ever invested in the mines and they are now idle.

Ore Knob, N. C.

THE ORE KNOB mine in Western North Carolina is the first of the important pyrites deposits northeast of Ducktown. Mining

was commenced about 1860. During the war the work languished. In 1873 the property was taken in hand by a strong corporation.

Large developments were made on the mine, and a complete Hunt and Douglas leaching plant was set in operation. Owing to the failure of the leaching process, smelting works were subsequently erected and have remained in operation until quite recently. The mine is a very long ore bed, with an almost vertical dip and the usual northeast strike. Eleven shafts have proved the bed over a length of 2800 feet. Extensive workings prove an average thickness of workable ore of about ten feet, varying from eight to sixteen. On the hanging or northwest wall a glassy quartz, carrying occasional patches of purple copper, is found, precisely as in the Mary mine of Ducktown, to which mine the Ore Knob bears great resemblance in its occurrence and in the character of the ore.

Ore Knob has been repeatedly described in the *Transactions* of the Am. Inst. of Min. Engrs. and in numerous private reports. The latest, a most complete description of the mine, works, and metallurgical process has been published by Prof. Thos. Egleston. From this report and other private advices I collate the following analysis and description. They will serve to illustrate the difference between the Ore Knob smelting practice and that of its prototype Ducktown.

The principal difference between the ores of the Mary and Ore Knob mines is the more silicious character of Ore Knob. Even this difference is not very marked. The metallurgical treatment of the ores is practically the same until after the smelting of the roasted first matte. From this point the treatment differs. At Ore Knob all second matte is returned to the roast piles, mixed with the first matte, and again roasted. Black copper is produced as at Ducktown, but refined without the interposition of the blister smelting. In the details of the treatment some differences and some improvements have been made at Ore Knob.

In ore roasting smaller piles of only 150 tons, made up of 50 tons of partially roasted, and 100 tons raw ore, and covering a floor space of 17 × 20 feet and 8 feet high, are used. The bottom is covered with fine ore and the whole pile is thoroughly covered with fine ore, and burnt under sheds. That the large loss of copper by leaching noted at Ducktown is thereby avoided goes without saying. Time and expense are also saved. The smaller piles become extinct in six weeks, and then at once, after only one fire, yield about 90

tons of ore roasted down to seven per cent. sulphur, which is sent to the smelter. Ore smelting at Ore Knob is done precisely as at Ducktown. Only one improvement has been made, and that is the use of crushed quartz for the bottom of the furnaces in place of steep. Longer campaigns result. The furnaces smelt rather more ore per day, reaching 13½ tons, owing to a larger volume of blast, furnished by a blower and not by a blowing engine as at Ducktown. The lighter blast must also be some advantage in preventing ironing. Campaigns last from 40 to 60 days. Owing to the better roasting of the ore and also to its superior quality, richer matte and a somewhat more basic slag result.

The mattes usually average 22 per cent., seldom dropping below 20 per cent. The slags run as low as 27 per cent. silica against 33 per cent. at Ducktown. Copper lost in the slag is claimed to be less than at Ducktown, the analysis by Prof. Egleston containing 0.57 per cent. and other analyses quoted by the managers even 0.4. There seems to be no good reason, unless it be the multiplicity of the bases in the Ore Knob slags, why copper should be lower in this more basic slag than similar slags at Ducktown. In Tennessee and elsewhere it has been the rule that more silicious and less irony slags are cleaner, as evidenced by the analysis of slag made from East Tennessee ore alone. Matte at Ore Knob is tapped in basins and removed in disks and then cracked, exactly as at Ducktown.

Roasting the first matte is carried out in piles of 30 to 35 tons on a paved floor and under cover. The piles measure 14 × 14 and are 5 feet high. Charcoal braize is used as a part of the bed and the usual amount of cordwood. Six fires reduce the sulphur to 4 per cent. Judging from the large amount of silicious residue shown in Prof. Egleston's analysis of roasted matte, the strictures made on the Ducktown process are applicable here as well. Indeed it is difficult to conceive from where so large an amount of insoluble material (13 per cent.) has been gathered, if only fair cleanliness in handling the first matte is observed. Casting first matte into pigs and crushing would be a decided improvement.

Matte smelting or rather smelting for black copper is carried out in furnaces like the ore furnaces, resulting in treating a like amount, or about 13½ tons of matte per day, with a production of about 7000 pounds metal—rather more than one-half black copper and the balance second matte of 55 per cent. As has been mentioned the

matte reverts to the roast piles and the black copper is sent direct to the refiner. This furnace is precisely like those of Ducktown, the only difference being that the bottom is seasoned with black copper and not with blister.

The charge of 8000 to 10,000 pounds is oxidized in the manner usual in the Swansea process and the skimmings drawn as necessary, the first skimming shortly after the melting, which has taken about six hours. Oxidation is then encouraged by rabbling and the skimmings are drawn every half hour or less during a period sometimes reaching twelve hours, or until the metal is practically a blister copper. Finally it comes to a quiet fusion and the copper is poled to tough pitch.

In this connection the accumulation of cobalt and nickel noted in the last skimmings, by the very complete analysis of Prof. Egleston, must be mentioned. The presence of minute traces of these metals in Ducktown ores had been noted by the writer, and its more marked presence in skimmings from the blister furnace. And it was owing to the evident concentration of these impurities in the slag, that it was returned to the ore smelting and not to the matte smelting. An instance of the liquation of silver lead from the black copper charged in blister smelting is likewise on record in Ducktown. It is doubtful whether the Ore Knob copper refined direct from black copper could be as pure as Ducktown first blown into blister. At any event the market favored Ducktown as against Ore Knob, although this may have been a part of the opposition still so widely manifested by consumers against any new brand.

The cost of mining at Ore Knob and at Ducktown were almost identical. Considering the higher price of labor at the last mentioned mines that work was done with greater economy in Tennessee. In the cost of smelting Ore Knob had not only the advantage of cheap charcoal and wood, at respectively 5 cents and $1.60 against 6⅔ cents and $3 at Ducktown, but the item of labor was also considerably reduced by reason of the compact nature of the Ore Knob Smelting Works and their proximity to the mines. Finally the ore smelted at Ore Knob yielded 100 pounds copper per ton against less than 80 pounds at Ducktown, a clear advantage of the value of 20 pounds of copper for every ton of ore treated.

Hence it cannot surprise that the Ore Knob works have continued in operation, and have even paid dividends, until the late unexampled drop in the price of copper. In common with the

other pyrites deposits that cannot make their sulphur available, this property is now idle, and in the hands of a receiver.

Within the last two years the Rio Tinto process has been introduced at Ore Knob, and some 10,000 tons of second class ore, containing an average of 2½ per cent. of copper, have been subjected to it. The receiver of the Ore Knob Company reports rather unsatisfactory results to date from the process, and lays the fault to a faulty bottom of the leaching yard. It would seem that this feature would give rise to trouble in any process like the Rio Tinto, requiring a great expanse of floors or bottom impenetrable to solutions. In this as in other considerations the process invented by the writer at Ducktown is superior.

Southwest Virginia.

The counties of Floyd, Grayson, and Carroll in Southwestern Virginia are the next in the order of our northward march to claim our attention. In Floyd County, the gossan of iron pyrites deposits has been largely used as an iron ore. Very little copper has been found, and the deposits probably have a character somewhat similar to those of Louisa County, Va., of which more anon.

In Carroll County beds are found very similar to those already described. A quartz hanging wall has here been noted, and the usual black ores were mined and smelted before the war. With the exhaustion of the decomposed surface ores the works languished, and finally collapsed, and are now amongst the many abandoned pyrites mines.

The thickness of the ore beds here varies from 20 to 70 feet. Their dip is usually at an angle of 45° to the south. McCreath, in speaking of them (*Am. Inst. M. E.*, 1884), supposes them to be a continuous bed, as much as 15 miles long. Further developments will prove this view to be erroneous. It was formerly very generally entertained in pyrites districts, but invariably abandoned as the mines progressed.

McCreath reports the Carroll County pyrites to assay 2½ per cent. copper and 45 per cent. sulphur, and the balance iron and gangue, which agrees with the results obtained elsewhere.

Louisa County, Virginia.

LOUISA COUNTY, VA., is traversed by the Chesapeake and Ohio Railroad. Where this road crosses the Lower Silurian slates at

Tolersville pyrites beds have been discovered. All the beds described heretofore were found in close proximity to gneiss rock, usually 50 feet or less from the walls of the ore body. Their age is, without doubt, Eozoic. The Tennessee mines have been so pronounced by Safford, although Whitney disputes this, and places the Ducktown ore beds amongst the Lower Silurian slates. With respect to the Louisa County, Va., deposits the writer admits the views of Whitney, but not as regards the other ore beds. It is probably to a later geological origin that the essential differences between the Tolersville and deposits described so far is due.

The shape of the Tolersville ore bodies is lenticular, as elsewhere. Their strike is about N. 25° E. and their dip 80° to the S.E. One bed, the "Arminius," has been prospected to a depth of 300 feet, and proved to have a width of at least 60 feet, and a length of probably 500 feet. Other beds have been developed north of the Arminius on the property of the Sulphur Mines Co. of Va. Four shafts have been sunk on this property, developing ore bodies extending over a length of four thousand feet. The average thickness is some 18 feet, the maximum 60. No great depth has been reached on the Sulphur Mines property at any of the workings.

In the character of their ores these deposits vary essentially from those already described. Little or no copper is found in the pyrites, and the bulk of this pyrites is no longer a Pyrrhotite, but a true pyrites or bisulphide of iron. Black oxides of inferior quality are found very sparingly below the cropping, and the sulphurets approach close to the surface, although the country rock is decomposed to a considerable depth. On the Sulphur mines property, iron ore was formerly mined from the gossan over a length of nearly two thosand feet.

Near the surface, and sometimes to a depth of 150 feet and more, the pyrites assumes a granular form, and can be mined without the use of powder or drill. In greater depth the granular ore ceases, and is replaced by hard compact pyrites. Parallel with the longer axis of the Arminius ore body there are streaks or zones of ore that contain a very appreciable percentage of galena. The galena occurs interspersed as fine crystalline grains throughout the pyrites. Ore in quantity can be selected with one-half the mass galena.

The Arminius mine was opened some ten years ago by a stock

company that worked the property for a short time. Soon the rich decomposed copper ore was exhausted, and the corporation ran into debt. Finally the property passed into private hands. For many years operations were suspended. The mine filled with water, and even the main shaft was allowed to cave in.

Recently a new lease of life has been given to the mine by reason of the gradual but certain introduction of pyrites in the manufacture of sulphuric acid. With unabated energy the owner of this mine had for years tried to sell the ore for that purpose, and our technical journals preached the economy of the use of pyrites in place of Sicily sulphur, but until recently without avail. Only within the last three years could manufacturers of acid be induced to even *try* pyrites. The *use* now firmly established, competition will do the rest, and force even unwilling and unprogressive manufacturers to adopt a cheaper raw material for their acid.

The Arminius mine has sold small quantities of pyrites for several years. The principal purchaser was formerly located near Baltimore. Within the last two years the demand has quickened sufficiently to warrant the reopening of the mines, which are now in operation and can deliver pyrites at Norfolk or Newport News at a freight-rate, from the mine, of $1 per ton. From that harbor most of the northern ports can be reached, by sail, for $1.00 per ton.

Placing the cost of mining at $2, an ore containing almost 50 per cent. sulphur can be laid down in New York for about $4.00 per ton, or 8 cents per unit of sulphur. Sicily brimstone, containing 97 per cent. sulphur, sells at tidewater for about $24 per ton, or about 20 cents per unit.

Pyrites burners can well afford to pay almost half that price, or say 10 cents per unit, and retain a liberal margin for cheaper production of acid. The oft-named drawbacks to pyrites, such as smaller produce for a given plant, greater cost of repairs, and nitre, etc., are more than offset at such a price.

In the Tolersville pyrites deposit we reach the first of those in active operation, and find these mines paying, notwithstanding their poverty in copper. As we continue our progress, we shall see that only such pyrites mines are in operation that make the sulphur profitable. The distance from railways, of the large southern deposits, is a bar to the beneficiation of the sulphur, and without the added value of this ingredient, the time has passed for the profitable smelting of any low grade pyrites.

Of the ore shipped from the Tolersville mines, the granular ore has proved to be the richest in sulphur, and it is now practically the bulk of the ore sent to market. An analysis of 49 per cent. is not unusual. The hard ore rarely exceeds 47 per cent.

The following complete analysis of Arminius ore has lately been published by W. H. Adams. It fairly represents the pyrites ore in all but its contents of lead:

Analysis by Dr. A. Volcker, of London, June, 1881.

Sulphur,	48.02
Iron,	42.01
Ferric oxide,	1.93
Sulphuric acid,	.44
Silica,	7.60
Copper,	none.
Arsenic,	none.

The copper-bearing portions of these ore beds contain less sulphur and more silica, and approach more nearly to the composition of the ore beds of Ducktown and Ore Knob.

Other analyses for sulphur made by Ledoux and Ricketts from samples drawn by W. E. Hildreth gave the following results.

Sample.	Sulphur.	Arsenic and Antimony.
1	50.20	none.
2	52.44	"
3	48.32	"
4	44.22	"
5	46.32	"
6	45.02	"
Average,	47.43 per cent.	

Sales are made on a guarantee of 45 units of sulphur.

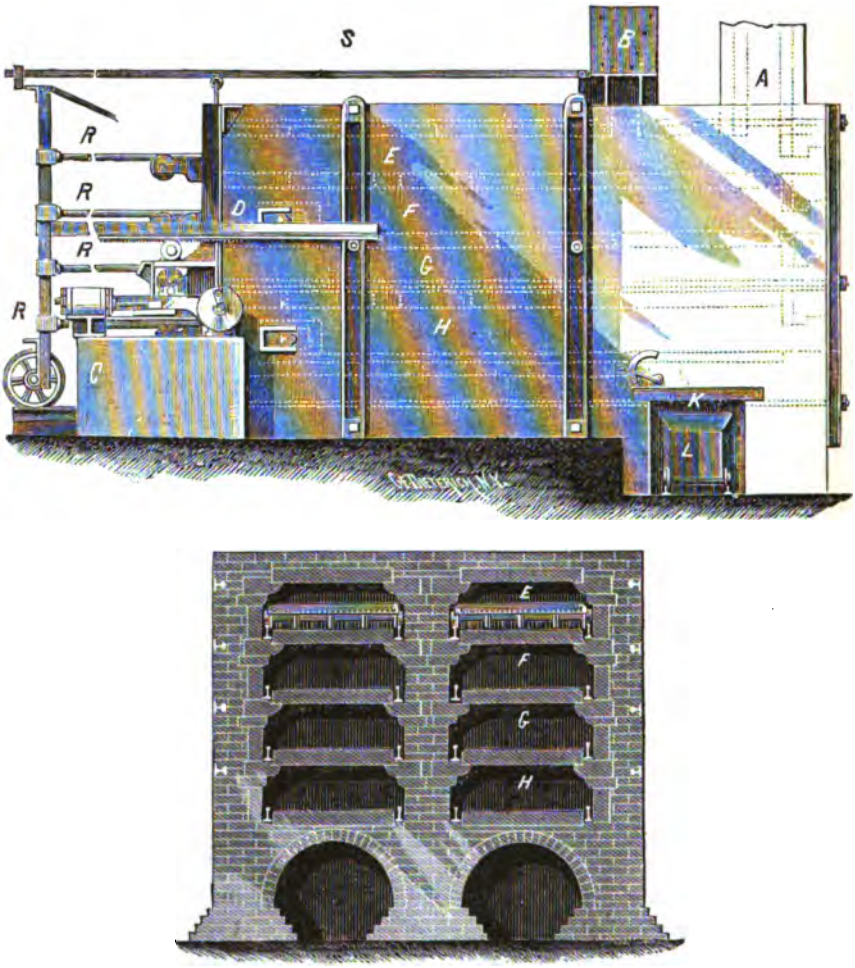
As has been stated, the bulk of the ores of these mines reach the markets in a granular shape, and were considered objectionable for that reason until the introduction of the Spence furnace.

This furnace is a 4-hearth reverberatory with mechanical feed and rabbles. It is in extensive use in Europe for smalls from the Tharsis, Rio Tinto and kindred pyrites ores. Practically the same furnace has been used for some years at the Mathiessen and Hegeler Zinc works for roasting blende. Granular Virginia ore which is both

richer in sulphur, and freer from impurities than European pyrites is very thoroughly burnt in these furnaces.

The woodcuts clearly illustrate the construction of Spence furnaces.

FIGS. 16 and 17.



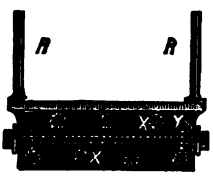
Spence Furnace.

The four floors, E, F, G, H, of the furnace carry the fine ore, which is charged at regular intervals and in stated quantities, through the hopper, B, by the rod, S, which is set in motion by movable lugs actuated by the wrought-iron frame, which carries and imparts motion to the rods, R, of the machine rabbles.

The frame is shown broken and not in full length, which equals the length of the furnace. The iron frame runs on wheels, and is set in motion by the rack, D, actuated by a pinion geared to the reversing engine set on the foundation, C. The gear-wheels are omitted in the cut to simplify it. The ore entering at B is evenly spread on the floor, E, by the upper rabble. Then it is transferred successively to the lower floors through the slots shown in dotted lines. Finally it is discharged in a roasted condition at K into the car, L.

The acid gases escape through the slots in a direction contrary to the travel of the ore, and escape through A into the chambers. A rabble is shown in detail in Fig. 18. It is moved by the rods, R,

FIG. 18.



before referred to, and is supported by the wheels, Y, running on the rails shown in the cross-section on each floor of the furnace. X are the prongs or teeth of the rabble, which can be replaced when worn. Depending on the speed of the engine a furnace will treat from 3 to 5 tons per day. If not pushed too hard, sulphur in the spent ore when treating Virginia pyrites rarely exceeds 1.25 per cent. and is often reduced to 1.0 per cent.

The lowest hearth of the furnace does but little work. On the one above the sulphur in the ore is already reduced to 8 to 9 per cent. The most active roasting takes place on the third floor.

With Spence furnaces and adequate lead chambers, 40 units of the sulphur of Virginia ores will be recovered as acid, that is one ton of ore will make considerably over one ton of sulphuric acid. It is for this reason that it is always economical to ship pyrites to the centres of consumption of acid, rather than to ship the acid. Not only can the ore be shipped at less cost per ton, but any other valuable contents of the ore are transported practically free of cost and can be recovered at less expense.

The Arminius and Sulphur mines of Va. companies both have branch tracks from the Chesapeake and Ohio to their shafts, and

the combined output of the properties at present is some 40,000 tons per annum, which amount can however be readily increased to meet any demand.

Again turning our face northward, we see the Eozoic rocks becoming less and less prominent; and we pass through the States of Maryland and Pennsylvania without finding any very notable deposits of pyrites. In Maryland, Cecil County claims to contain some beds, of no great value; and, with the exception of the Gap mine, in Lancaster County, Pennsylvania, no pyrites beds of magnitude are found until we reach the Hudson. The Gap mine pyrites is valuable only for the nickel it contains. The bold promontory of Anthony's Nose, in Putnam County, New York, has been the seat of a mining enterprise now abandoned. A bed of pyrites is found there some twelve feet wide, but much mixed with hornblende and other rock. After a careful assorting, the ore assays only 25 per cent. of sulphur, and no copper or other valuable metal. The ore was formerly carted down the mountain to the Highland Chemical Works, on the Hudson, a large acid plant, built with a view of utilizing this deposit. One hundred kilns were kept in operation at the acid works, and were daily charged with 75 tons of raw ore. The kilns carry a column of ore 3 feet high, and had no grates. If properly attended and allowed but little draft, they gave very satisfactory results, reducing the sulphur to 5 per cent. They were put in operation in the usual manner, by filling them two-thirds full, then building a strong fire on the raw or spent ore, and finally covering it with fresh ore. Once ignited, they remained continuously in operation. Fifteen hundred pounds of clinker were drawn once in twenty-four hours from each kiln, and as much fresh ore, broken to about three-inch cubes, charged through the side-doors. Owing to the low contents in sulphur, only twenty units, or the equivalent of 400 pounds of sulphur, were extracted from a ton of ore; and, after an extensive trial, it was found to be cheaper to use Sicily brimstone.

This is not surprising, for it must be borne in mind that if there is a profit in burning a 50 per cent. pyrite at, say 10 cents per unit, the real value of a 25 per cent. ore is not 10 cents per unit, but considerably less. The expense of handling a ton of ore remains the same no matter what the produce, and in the above example this produce was diminished in the ratio of 40 to 20, or one-half.

Ore so low in sulphur cannot, under present conditions, be made profitable without the added value of some contents in metal.

(To be continued.)

THE THICK COAL OF SOUTH STAFFORDSHIRE.*

BY HERBERT W. HUGHES, F.G.S.

* * * * * * * * * *

DESCRIPTION OF THE SEAM.—The "Thick" coal is composed of a number of beds of coal, varying from 8 to 14, resting either directly upon one another or separated by thin beds of "clunch" or "shale," called "partings." Each of these beds of coal has distinct lithological characters, and can thus easily be recognized by an old Thick coal miner and referred at once to its position in the seam. These beds have received distinctive names peculiar to the parts of the district in which they are worked; for instance, the lower bed is variously known as Humphreys, or Omfray floor, benches, holers, dice coal, kit, or kid coal, and red coal; the Roofs is sometimes called the top floor; the Spires is the top slipper; the White coal is divided into the Jays and Lambs, and sometimes, where this division occurs, the lower portion is called Lambs and the top portion White coal (Fig. 2); the Veins is the Slips or John coal; the Fine coal is the Foot coal, this being sometimes subdivided into top and bottom foot-coal; and the Heath coal is frequently called the Tow coal. A coal called the "fine floors" very often occurs between the "White coal" and "Heath coal;" this is probably the "nine-inch" of the Brettell Lane district.

* * * * * * * * * *

At the Barn pit, Parkhead, about one mile southwest of Dudley, the Thick coal has the section shown in Fig. 1. We may consider the coal at Scots Green and Parkhead, to be not only the best in the district, but the finest seam in the world, for all around here it measures nearly 11 yards of clear coal with only a few inches of

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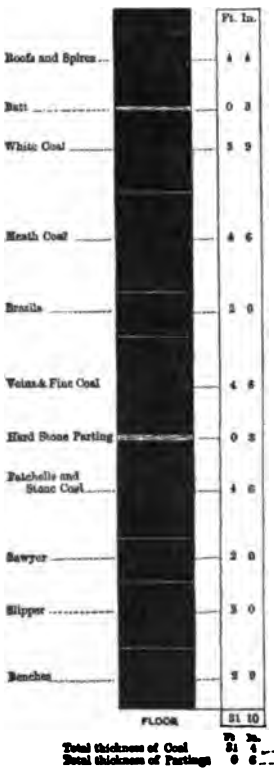
* Extracts from a paper read before the British Society of Mining Students, August, 1885, by Herbert W. Hughes, Dudley, Worcestershire, England.

partings, and, moreover, is of excellent quality; unfortunately it is now nearly exhausted. About one mile further on in the same direction, we find at the Earl of Dudley's No. 25 pit, Wallows Colliery, the coal having a clear thickness of 11 yards 2 feet 3 inches with 9 inches of partings (Fig 4).

* * * * * * * * * *

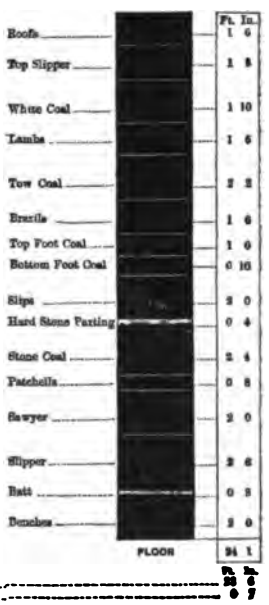
Fig. 3, extracted from Beete Juke's work on the Geology of the South Staffordshire Coal-field, gives a diagrammatic representation

FIG. 1.


Ft. In.
Roofs and Spires 4 4
Batt 0 3
White Coal 3 9
Heath Coal 4 6
Brazils 2 0
Veins & Flint Coal 4 8
Hard Stone Parting 0 3
Patchells and Stone Coal 4 6
Sawyer 2 0
Slipper 3 0
Benches 2 9
FLOOR 31 10
Total thickness of Coal 31 4
Total thickness of Partings 0 6

Barn Pit, Parkhead, near Dudley.

FIG. 2.



Sandwell Park Colliery, West Bromwich.

of the splitting up of the Thick coal and other coals towards the north, the following explanation being there given: *

"Each section is a mean or average section deduced from a

* J. Beete Jukes, *Geology of the South Staffordshire Coal-field*, p. 25.

FIG. 3.

Thick Coal
Bottom Coal
Brooch Coal
Bottom Coal
Brooch Coal

considerable number of actual pit sections. The coals known as the Heathen and Sulphur are common to the two sections, the Heathen being assigned as the horizontal datum line. The Fire-clay and Bottom coals are also common to two sections, being worked continuously from Bilston to Bently, and thence to Pelsall.

"Just at the south end of Pelsall Heath, the Bottom coal hitherto twelve feet in thickness and worked continuously as a single seam, begins to separate into two coals, which about a mile or so further north are 45 feet apart.

"The Fire-clay coal having been worked continuously from Bilston to Pelsall, begins to change its quality and character, and further north is known only as the Cinder coal.

"The New Mine coal crops out before it reaches the Great Bentley fault, and when thrown in by that fault is found to be separated into two, called at Bentley the Three-foot and Five-foot, but known further north as the Yard and Bass coals.

"With this tendency to split up into separate coals as we proceed towards the north clearly proved in the lower beds which run continuously in the ground, we can the more readily allow the probability of a similar tendency in the upper beds, and the general facts of the beds at Essington, Wyrley, and Bentley, from the Old Robins coal down to the Bentley Hay coal inclusive, being the representatives of the Thick coal can no longer be doubtful."

The Thick coal is classed with the non-caking bituminous varieties, and it is the first departure from coals of the Lignitic type. In conducting chemical analysis to determine the uses for which it is favorable, I consider it extremely necessary that the composition of every seam in the bed should be determined separately, as each one varies in quality from the others. The following are some results given by Dr. Percy,* in which attention has been observed on this point:

* Metallurgy, *Fuel, etc.*, 1875, p. 325.

Number.	Name of Seam.	Composition, per cent., exclusive of Water only.					Composition, per cent., exclusive of Sulphur, Ash, and Water.		
		Carbon.	Hydrogen.	Oxygen.	Sulphur.	Ash.	Carbon.	Hydrogen.	Oxygen and Nitrogen.
1	Roofs.....	76.12	4.83	16.72	1.00	2.33	78.46	4.96	16.58
2	Top Slipper...	77.01	4.71	16.72	0.74	1.56	78.53	4.80	16.67
3	White coal....	76.40	4.62	17.43	0.55	1.55	77.68	4.69	17.63
4	Brazils.....	72.13	4.32	17.11	0.54	6.44	77.33	4.67	18.00

* †

"The coals were from the Whyley Colliery, West Bromwich. The powder of each of these coals is brownish-black. The color of their respective ashes is as follows: No. 1, reddish-gray; No. 2, yellowish-red; No. 3, yellowish-red; No. 4, reddish-gray. The portion analyzed of each coal was dried somewhat above 100° C. Combustion was effected in oxygen."

A short time ago, I tested a specimen of White-coal from the Earl of Dudley's Himley Colliery, and obtained the following result:

Fixed carbon, ...	52.50 per cent.
Volatile matters, ...	32.14 "
Ash, ...	2.26 "
Moisture, ...	11.19 "
Sulphur, ...	1.91 "

The color of the ash was light buff. Calorific power, determined by Thompson's Calorimeter, proved that one part of coal would evaporate 14.025 parts of water into steam at a temperature of 212° F. The evaporating power of Welsh steam coals being about 19, this may be considered a fair result, its lowness being due to the large amount of moisture these Staffordshire coals contain.

DRIVING OUT GATEROADS.—After the shafts have been sunk and the pit bottom secured and laid out, the gateroads are driven out to the boundary; the invariable rule being, whatever system of getting the coal is applied, to first reach the boundary and then bring the work back. Of course if you have a large area, and the

* The Nitrogen is included in the number indicating Oxygen.

† Calculated after deductions for oxygen present as sulphuric acid, oxygen present in iron in ash, and also correction made in the ash.

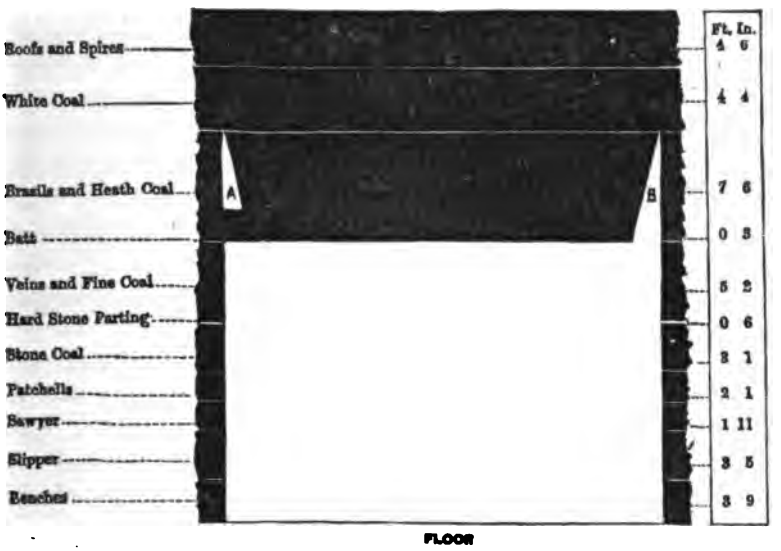
main gateroads are driven along the "strike" of the seam, they may serve for haulage roads; side roads can then be driven out to the boundary in the direction of the rise, and coal can at once be gotten at the extremity of these latter, even before the former have proceeded much past them. The only precaution to be observed is that the coal so worked shall be at sufficient distance from the shaft so as not to affect it by any subsidence. While this portion is being worked out, the main roads proceed in their course, and at a suitable distance, side roads are again sent out to the rise, and, when they reach the boundary, work is opened as before. The exhaustion of the mine in these side roads, is never allowed to proceed to such an extent as to throw any weight on the main haulage road, for before that point is reached, fire dams are put in, and work abandoned there. This system of laying out the pit offers many advantages; in the roads driven along the strike one of the methods of mechanical haulage can be adopted, and this can be supplied by self-acting endless chains working in the subsidiary side roads.

The above plan can only be completely carried out when the shafts have been sunk in the deepest part of the mine. When there is a certain area to the deep, the roads are driven out along the line of greatest dip, if it be a moderate one of say 10° to 15°,—if greater, they will be driven diagonally. When these roads have reached the boundary two other gateroads are driven along the strike, and work opened at their extremity; the produce being brought along these latter roads, and then hauled up the dip roads by means of a rope, several tubs being brought out at a time. Owing to the steep inclination the empty tubs will generally run in by themselves; I have only seen one instance in which a tail rope was applied for this purpose.

Gateroads are driven about 9 feet wide, and when completed will be about 6 feet high, if the benches, slipper, and sawyer are taken out. Holing commences in the benches, and that seam, with the benches batt, marks the height cut away in this portion of the work. When the holing has proceeded for about 15 feet under (the roof in the meanwhile being supported by short pieces of timber called "puncheons"), two or three workmen, at distances of 6 feet measuring from the rear of such holing, proceed to cut vertical grooves up through the overlying measure of coal leaving between each length of six feet, what are called "spurns." These spurns are narrow webs of coal; a view of them in elevation is given in Fig.

4. **A** shows the spurn in, and **B** after it has been removed. These spurns are holed through in the upper part. When the slipper and sawyer have been cut through in this manner, sometimes on both sides, generally on one side, the spurns are reduced by the aid of a pick, and are then finally "jobbed" out with a "pricker," which is a long instrument very similar to a boat hook. A spurn is always left at the face, and when this is removed the whole mass falls, and is then in a position to be taken away by the loaders; after this has been done the uncut side is dressed straight, and the road is then

FIG. 4.



Scale 12½ feet to 1 inch.

fit for timbering where needful. Sometimes the patchells is cut in an arched shape, timbering thus being rendered unnecessary.

The nature of the roof and floor varies considerably in different parts of the coalfield. A gateroad in the Parkhead coal would stand years where one in the West Bromwich coal would only stand months; in view of this latter fact, we drive the gateroads at Sandwell park Colliery with the sawyer as floor and the Brazils as roof, the slipper, sawyer and benches being thus left under foot in the gateroads, thereby preventing any risk of creep.

It was formerly the custom to drive air-heading alongside the one gateroad going out to the boundary (the roof of the air-heading

being driven on a level with the roof of the gateroad); cross cuts, called spouts, were put through into the road and thus ventilation was carried out. The air-heading was driven 4 feet wide by 4 feet high, and as its roof was on a level with the roof of the gateroad, it effectually removed any accumulation of carburetted hydrogen.

The general practice is now to drive two parallel gateroads, reserving air-heading for the purpose of ventilating sides of work. By a modification of the old method of working, even this small quantity of air-heading is now done away with, and this arrangement will be described in the next section when treating of the various methods of working.

VARIOUS METHODS OF WORKING.—(*a*) *Square work.*—After the gateroad and air-head (A and B, Fig. 5) have reached the boundary of mine to be worked, the first operation in proceeding to win the coal by square work, will be to drive a narrow head called a bolt-hole (C, Fig. 5). When this has reached a distance of from 10 to 15 yards, its width is increased to 6 or sometimes 9 yards, then going on as the stall D. In driving these stalls, the first operation is to hole the benches about 9 yards under, the roof being supported by small cogs of slack, or by short props. The slipper and sawyer will then be vertically cut on both sides of the stall, spurns being left at intervals of 6 feet, as in the case of gateroads. When these spurns are removed, it frequently happens that the patchells and stone coal fall with the slipper and sawyer; if this occurs, the sides are dressed straight and timbering put in to support the roof. Further lengths of 9 yards are similarly attacked in the stall C until the back rib is reached. While this has been going on, other stalls, E, F, G, and H have been driven in the directions shown by the dotted lines, and finally, the pillars will be thirled by means of the short lengths shown at K, L, M, N, O, and P, in the same figure; after the driving of the stall R, the side of work is fully opened, and cutting in of the top coals will take place.

The cutting in of the top coals proceeds in the same manner as previously described when treating of gateroads, each layer or bed of coal being removed in ascending order. If the roof is very good a long range can be taken, but it is far better to do this part of the work in short ones; distances of 20 yards are taken, and a timber cog built up to the roof. A vertical groove will then be cut on each side of the stall and on the face of the cog, treating the latter as if

it were solid coal, spurns being left as before; their appearance when the Brazils and Heath coal are being cut is shown in Fig. 4. When these spurns are taken out, as shown at B, and the timber removed, the whole mass within the portion cut falls down and is ready to be loaded up.

Similarly a layer of top coal is removed all over the side of work, and when this is done, the white coal, and then the top slipper, will be attacked, and the same cycle of operations gone through until these seams have been cut in. No attempt is made to get the

FIG. 5.

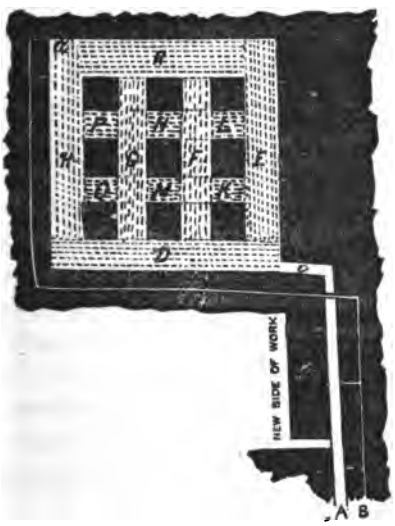


FIG. 6.

Scale 132 feet to 1 inch.

"roofs;" sometimes, however, they break down without the superincumbent strata following, in which case they are removed as speedily as possible. In cutting the top coals, the workmen stand on a platform constructed by cutting two holes in the solid coal, into which short timbers ("byatts") are inserted, on which a plank is placed; preferably two poles are inclined against the side, to the tops of which pieces of chain are lashed, and a loop made to receive each end of a plank.

The above describes how a side of work would generally be exhausted, but numerous modifications are carried out in special cases.

A very frequent proceeding is to build timber cogs up to the roof when the bottom coals have been removed; these cogs are placed in the openings between the solid pillars of coal, and when this is done the size of the latter is reduced in a marked degree. If the reader will refer to Fig. 7, he will observe the pillars are not quite so square as in Figs. 5 and 6. Fig. 7 has been copied from an actual working plan, and shows how the pillars are reduced before the side of work is finally abandoned; the small cross shaded parts represent cogs.

It frequently happens that the top measures are not removed all over the side of work at once; the bottom coal is not of such good quality as the top coal, and it may, therefore, be necessary to mix some of the former with the latter. In this case, while the stalls are proceeding in the bottom portion, the top coal will be cut in all along the back rib and mixed with that obtained in driving the stalls. The top seams may be cut in until the roof is reached and the shut comes down. Timber cogs will then be built in the openings nearest the shut, the solid pillars removed, and the top coal cut in the next range all around these artificial pillars, thereby greatly increasing the yield. The top coals repose on the shut, and thus a large amount of weight is removed from the cogs; these cogs are taken out, rebuilt in the next series of openings, the solid pillars in this range are removed, and the top coals cut in here. Of course the roof will fall in behind these cogs and again afford support for the top coal. By this procedure a greater first yield is obtained, as the pillars are sometimes completely removed, and in many cases much reduced, as will be seen by referring to Fig. 7. If the seam is very liable to spontaneous combustion, or has an unsuitable roof, this modification cannot be carried out, although it would obviously be advantageous in every case.

In the old method of working, the ventilation was effected by air-heading, and the course of the air can be easily traced by referring to Fig. 5; the back spout *a* is sumped up as the top coals are cut in.

The air-heading used in ventilating the gateroads and sides of work, can be dispensed with by using the system shown in Fig. 6. Here two parallel gateroads A, A′, are driven from the shaft until the boundary is reached. The back opening D will be thirled, and then the gateroads widened into stalls, by means of "side lanes," as shown at B and C. The pillars can be thirled as before, indi-

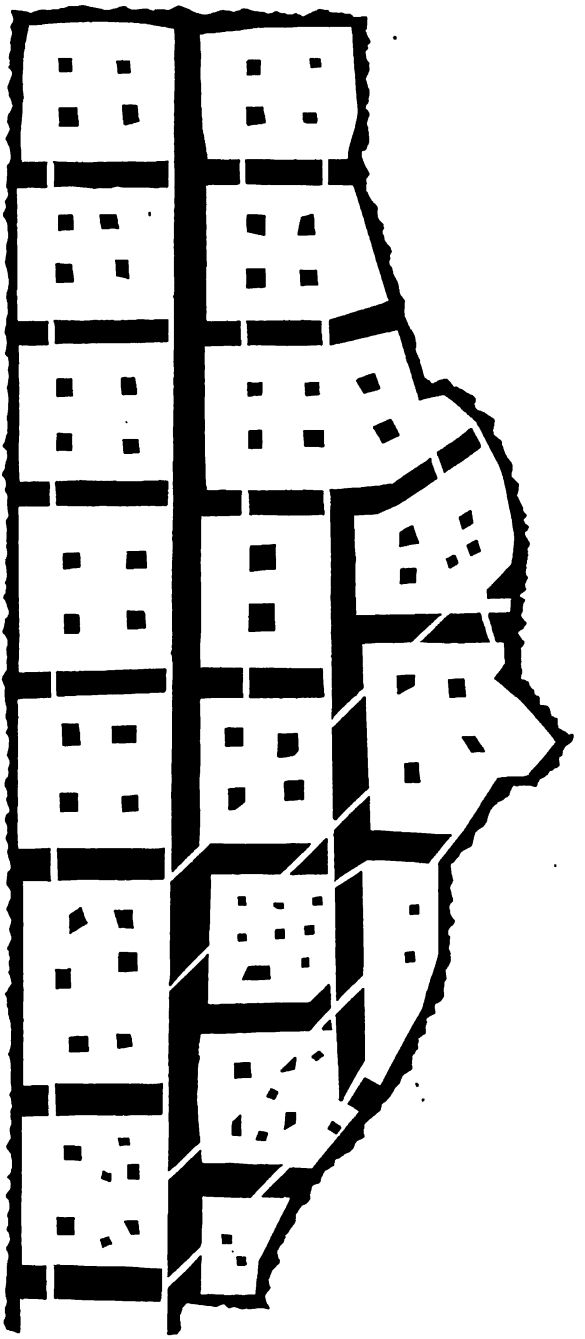


FIG. 7.

Scale 191 feet to 1 inch.

cated by dotted lines, or the back opening can be cut in and timber cogs placed between the pillars in the openings F and G. This method of laying out the work is now very generally adopted, and was the one used in working the area shown in Fig. 7; the course of the primary gateroads will be easily followed on this plan.

When the top coal has been removed in a side of work, the pillars commence to crack and split; supposing the roof to hold up, a quantity of good large coal can easily be obtained, by "robbing" the pillars. Slices are cut off them with as much speed as possible, and they are worked at until the cracking of the roof warns the collier of his danger. When the roof falls in, a dam built of refuse is placed in the bolt hole, and a good brick and mortar wall placed on the gateroad side. The brick dam is placed deep in the floor, roof, and sides of the bolt holes, and is intended to hermetically seal the side of work just finished; this damming is done immediately the last coals are drawn out, otherwise spontaneous combustion would occur.

While the drawing of the top coals is going on in the first side of work, preparations are being made for opening another one in the bottom coal, leaving between the sides of work, a rib of coal about 10 yards wide (Figs. 5 and 6), and by the time the first side is exhausted, and the fire-damp put in, the second one will be in full working order.

(*b*) *Longwall* (*the seam being removed in two slices*).—At several collieries the thick coal has been won at two operations by the longwall system, or what might preferably be called a "longwall modification of pillar and stall." The top coal is always cleared away first; gateroads are driven out to the boundary and rectangular blocks of coal formed. Working homewards is always practiced. After sufficient time has elapsed, the working away of the bottom coal is effected in a similar manner.

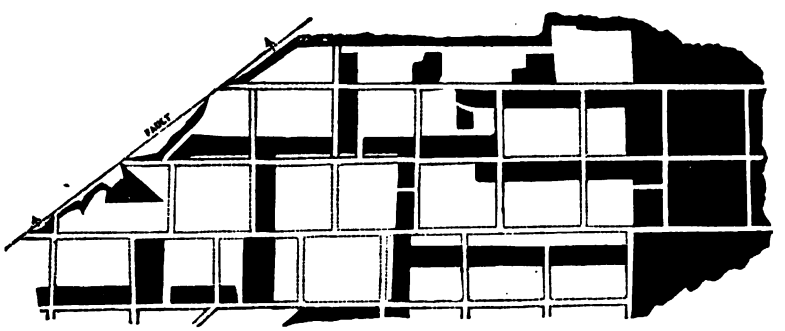
When first introduced, great advantages were claimed for this method by its adopters, who at once said this was the proper way to work the seam. There can be little doubt that some thick coals can be worked very advantageously by this method, but I think it will be admitted that the South Staffordshire Thick coal cannot, when it is mentioned that after 20 years' trial, this system has been everywhere abandoned. There are several objections against it; you get a greater percentage of slack, and as was mentioned be-

fore, the top part of the Thick coal is of far better quality than the lower portion, consequently the owner has no difficulty in selling the top coal, but while the lower portion is being worked he has the greatest difficulty in so doing, and probably has to dispose of the produce at a lower rate. Another great disadvantage is the amount of gob-fires one has to contend with. Probably these fires occur more frequently in the Thick coal than in any other seam in the world, and immediately a fall occurs in the roof, when working the bottom part of the seam, one of these outbreaks is encountered, and follows behind the receding face with great pertinacity, every fresh fall in the roof only increasing the danger.

The above are probably the reasons why the system has been abandoned; it will be shown later, that although the yield per acre is greater by this method than by the preceding one, yet when we consider the question commercially, there is very little to choose between them, owing to the greater quantity of slack obtained when working on the longwall system.

(*c*) *Longwall* (*the whole thickness of coal being removed at once*).—This, the last method of working to be described, was suggested in

FIG. 8.



Scale 264 feet to 1 inch.

a slightly different form by T. Smith, in 1836. Referring to Fig. 8, it will be seen that the area to be worked is subdivided into rectangular blocks of coal by gateroads, which intersect each other at right angles. It is the object of the miner to clear out these blocks, and various methods of procedure are adopted in doing so. Instead of calling the work "longwall," we might preferably define it as "pillar and stall," the large blocks being considered pillars; if we

do so, the system of working is exactly the same as the one pursued under the same title in the northern coal-field of England, and the working away of the pillars is similar, with slight modifications occasioned by the greater thickness.

In clearing out one of these blocks, a slice is taken off in the bottom coal parallel to the shorter axis, and the top coal placed on cogs. Another slice is taken off by the side of the former one, and similarly placed on cogs. The cogs in the first range will now be withdrawn, and the top coals removed until the roof falls in. A third slice will then be taken parallel to the second one, and cogged; when this is done, the cogs are removed from slice No. 2 and the top coals brought down here. It will be seen that the shut having fallen in slice No. 1, the top coals in slice No. 2 repose on this refuse, and on being cut in will roll down into the openings, and can easily be withdrawn. When a fire occurs, a rib is left of sufficient thickness, and then the removal goes on as before, more slices are taken off until the whole pillar has been worked away.

Another method of procedure, is to split the block along its longest axis, and place the top coal on cogs. Slices are then removed on both sides, parallel to the shorter axis. Sometimes the block is subdivided into four pillars, these pillars being again divided into four, and then each small pillar removed, commencing near the shut, and ending with those nearest the solid. The shut follows the extraction, and all through the working the top coals repose on it, and when these coals are cut in, they fall down the inclined surface of refuse into the openings where the men are at work.

For the success of this system, it is necessary that the coal should have a soft roof, and one that comes down quietly without much weight. In some parts of the coal-field there is a very hard roof which will bear a large amount of weight without collapsing; however—in the words of the collier—"when the weight does come on," nothing can stop it. Such a roof is very unsuitable for this system.

The advantages claimed are: the greater first yield, and total clearance of coal. The disadvantages are: the large amount of slack produced (this being due to the quantity of gunpowder employed), and the smaller *total* yield per acre. A greater quantity is obtained per acre, than by the *first* clearance in the square

work system, but, after the lapse of some considerable time, the ribs and pillars left in this latter method of working, are cleared out. In many cases the remnants of the Thick coal are worked a *third* time, thus obtaining a further yield, and will yield about one-third the quantity obtained by the first working, of which one-third will be coal and two-thirds slack; the expense of winning this broken mine is somewhat greater than that of getting the solid coal.

YIELD PER ACRE PER FOOT THICK OF COAL, BY THE ABOVE METHODS.—In calculating the yield per acre by the various methods of working, the quantities have been reduced, by dividing them by the thickness of coal cleared, thus obtaining an equal comparison. In the square work system, the quantities obtained by the second working have been added to those obtained by the first clearance, so that in all instances the coal is supposed to be totally exhausted. Further on a commercial comparison is given, in which the price of coal is taken to be 10/-, and the price of slack 4/-; the question of the amounts of large coal yielded by the various methods is of the utmost importance, for it is upon this variety that the owner looks for profit, the money cleared from slack and small coal being very slight, or, in the majority of cases, may be considered *nil.*

(*a*) *Square work.*

Area worked,		10.5 acres.
Thickness of coal gotten,		24 feet.
Quantity of mineral obtained per acre,	Coal	18120 tons.
	Slack	11713 "
Quantity obtained per acre per foot thick of coal,	Coal,	755 "
	Slack,	488 "
Total per acre per foot thick,		1243 "

(*b*) *Longwall* (*two divisions*).

Area worked,		18.925 acres.
Thickness of coal gotten,		23 feet.
Quantity of mineral obtained per acre,	Coal,	15219 tons.
	Slack,	16933 "
Quantity obtained per acre per foot thick of coal,	Coal,	661 "
	Slack,	737 "
Total per acre per foot thick,		1398 "

(*c*) *Longwall* (*one division*).

Area worked,		21.694 acres.
Thickness of coal gotten,		17.5 feet.
Quantity of mineral obtained per acre,	Coal,	7338 tons.
	Slack,	12879 "
Quantity obtained per acre per foot thick of coal,	Coal,	419 "
	Slack,	736 "
Total per acre per foot thick,		1155 "

Assuming the value of coal to be ten shillings per ton, and the value of slack to be four shillings per ton, we find the amount yielded by the sale of the mineral from each system of working (per acre per foot thick of coal) to be as follows:

	L.	*s.*	*d.*
(*a*) Square work,	475	2	0
(*b*) Longwall (two divisions),	477	18	0
(*c*) Longwall (one division),	356	14	0

These figures, and the preceding ones, speak for themselves. The great drawback to the two last methods, is the large amount of slack obtained; on an average the square work method will produce from fifteen to twenty per cent. more round coal than either of the other two systems.

I have estimated the area of thick coal remaining in South Staffordshire at 3050 acres, and as the quantity originally present was about 30,000 acres, we see that one-tenth of this famous seam still remains unworked. There is also little doubt in my own mind that a considerable area of thick coal is still unproved, and that for many years to come the thickest seam of coal in England will deliver up its stores for the use and benefit of the district.

* * * * * * * * * *

SPONTANEOUS COMBUSTION.—The occurrence of spontaneous combustion in the thick coal is somewhat frequent, the first signs being given by that peculiar smell termed "fire-stink;" this smell is exactly similar to that obtained when coal is distilled in a glass retort at the lowest temperature at which decomposition takes place. Combustion is considered to be caused by three actions, viz.: (*a*) Oxidation of iron pyrites, (*b*) Oxidation of the organic constituents of coal, (*c*) Friction from slippings, each of which will be considered separately.

(*a*) *Iron Pyrites.*—This substance on decomposing yields, first, ferrous sulphate, and, secondly, ferric sulphate; the former makes

its appearance in the form of colorless fibres protruding here and there from the face of coal, while the latter is of a brown color, and is more frequently observed. These products may suffer further decomposition—sulphuric acid being sometimes formed—and as their volume exceeds that of the original pyrites, disintegration of the coal is effected, together with a small heating of the coal in close proximity to each lump of pyrites. The heat generated is quite incapable of commencing a fire, but it may help in a great degree the action of other agents. Ferric sulphate is reduced to ferrous sulphate by contact with small particles of carbon, and hence may act as a carrier of oxygen to the organic constituents of coal.

(*b*) *Oxidation of the Organic Constituents.*—Richter's* experiments have satisfactorily demonstrated the high importance of this action, and it may now be looked upon as being the most effective of the three. Coal absorbs oxygen, one part combining with carbon and hydrogen forming carbonic acid and water, while the other enters into combination with the coal and proportionally increases its weight. This alone would fix our careful attention on this action, for we find that before combustion, coal so inclined emits large quantities of carbonic acid gas. Heating results from the absorption of oxygen, and absorption is favored by heating, moisture, fine division, and absence of light; everything thus combines to favor decomposition.

(*c*) *Friction from Slippings.*—Pressure from the roof on pillars cracks them, and grinds the irregular sides of these fissures together, thus producing considerable heat. Really solid pillars never fire, it is only on completion of a side of work, when the pillars are being crushed, that combustion occurs. The heat acting on small particles of coal produced by the grinding action, may subject the same to the process of slow distillation, and produce a quantity of bituminous matter, which may, on the addition of further heat, suddenly burst into flame.

Development.—The oxidation of iron pyrites cannot be looked upon as the primary agent in producing this undesired state of things. We can readily estimate by means of the calorimeter the amount of heat that would be given out by the decomposition of the quantity of sulphur in any coal, and on doing so can at once

* Dingler's Polytechnisches Journal, 1870, vol. cxcv., pp. 315 and 449; vol. cxcvi., p. 317.

convince ourselves that this could not produce the result attributed to it, even if the pyrites existed in isolated nodules. Another argument in favor of this is the very slow nature of the process, the heat produced is consequently dissipated, and only a small heating of the particles takes place. There can be little doubt that the decomposition of iron pyrites is eminently favorable to spontaneous combustion, owing to the disintegration produced thereby; it thus renders the coal in a fit state, more permeable to currents of oxygen; and by the heating produced, small though it may be, favors the action of such currents.

When we turn to the second agent everything seems to combine to render the action successful. Heating, moisture, and absence of light are all conducive to the oxidation of the organic constituents of coal; it is in seams most free from pyrites that spontaneous combustion takes place. The constitution of the coal seems of great importance, it is only in bituminous varieties that we find this undesirable state of things.

Little can be added to what has been already said on the heat produced by friction; the principal argument in favor of this view is the one already given, viz., if fire be found anywhere, it will be in the cracks of pillars. No doubt this is perfectly true as regards underground fires, but we frequently get spontaneous combustion occurring in heaps of coal above ground, and this coal may contain a very small percentage of pyrites. To account for the fire under these circumstances is impossible, unless the oxidation theory is admitted, and it is my opinion that in all cases this action is the primary agent, although either of the other two in conjunction may greatly facilitate it.

Prevention.—This can only be done by the loading up and removal of all fine slack and refuse. Small coal does not take up more oxygen than lump coal, but the absorption is more rapidly effected, and on this account coal dust is more inflammable. A vigorous current of cool air must be circulated through the workings, thus cooling the surface of coal over which it sweeps. The general practice in this district was until lately to reduce the quantity of air passing; thereby the risk of combustion was increased, because sufficient air is always left for oxidation, and owing to the small volume the air gets heated higher than the surrounding strata, and consequently aids instead of impedes the risk. There is certainly a point at which a vigorous current of air would be

unadvisable. If combustion *has* broken out, the quantity of air should be reduced, but until that point is reached a diminution in quantity would only act detrimentally.

As a cheap and effective method of preserving coal heaps on the surface, the system proposed by Wenzel Poech, of Karbitz,* is deserving of special attention. He admits spent steam into the piles, a number of trenches being dug for the purpose. These are covered with beams and boards, leaving narrow spaces. On these boards the coal is deposited, and the trenches connected with the exhaust-pipe of a steam-engine. For medium-sized coal, with a pile three yards high, the distance between the channels is three yards. The exhaust steam of a 4 horse-power steam-engine, worked for six hours per day, was enough for the preservation of a heap containing twenty trucks of coal. The cost of construction for a heap of 100 trucks of coal is about £3, and the cost of working is slight. The process has been employed with success at several works, including the Maria Antonia colliery at Karbitz, Ferdinand pit at Teplitz, and the Bruno colliery at Weisskirchlitz, near Teplitz.

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THE PARTING PROCESS USED IN THE UNITED STATES MINT AT PHILADELPHIA.

BY T. EGLESTON, PH.D.

THE United States Mint at Philadelphia is a fire-proof building, containing besides the works of the mint, properly speaking, large vaults for the storage of bullion and coin, and a museum of great historical interest. A parting process, invented in the year 1866, by Mr. A. Mason, the present superintendent of the New York Assay Office, but then its melter and refiner, is used for separating the precious metals. The same process is also used in the mint at San Francisco. It is carried out in both places very efficiently and cheaply. This process is known as the double

* Oesterreichische Zeitschrift für Berg- und Hüttenwesen, vol. xxxii., pp. 308, 320.

process, as both nitric and sulphuric acid are used in it. It consists in inquartating the bullion, so that the gold shall constitute 100 parts in 285 of the granulated metal; this proportion leaves sufficient margin for any want of homogeneity that might occur in the granulated bullion, while the porous character of the resulting metal causes the acid to dissolve the silver more completely, so that not more than 6 per cent. of silver is left in it. The residuum from the nitric acid is boiled twice with sulphuric acid, and makes gold 998 fine. This process is applicable to all kinds of alloys, and has been successfully worked since its invention. It is much cheaper than the old nitric acid process, requiring very much less fuel and 20 per cent. less nitric acid. The sulphuric acid part of the process is also very cheap, and requires but little apparatus and few men.

A brief summary of the work of this mint during the year 1884–1885 is given below. There were coined during the year $18,509,280.25 of gold and silver. This value was contained in 42,864,328 different coins. The amount of gold coined during this time was 8,880,322 ounces, with a wastage of .002 of a standard ounce. The amount of silver treated in the same time was 16,330,992.87 ounces, with a wastage of 2148.83 ounces, 1980.12 ounces of which were stolen, so that the actual wastage of the year was 168.71 ounces. The amount of gold and silver parted was $3,381,962. The weight of the gold in gross ounces was 288,747, in standard ounces it was 153,390. Its value was $2,853,766. The weight of the silver in standard ounces was 453,919, and its value was $528,196. The charges collected from depositors for the parting were $12,188.02. The expenditures for doing it were $8250.97, leaving a profit of $3937.05.

The process used consists of eight operations. These are:

1. Receiving and melting the deposits.
2. Inquartation and granulation of the material containing gold.
3. Solution of the granulations in nitric acid.
4. Boiling the gold residues twice in sulphuric acid.
5. Precipitation of the argentic nitrate with salt.
6. Reduction of the argentic chloride with zinc.
7. Sweetening and pressing the silver into cakes and drying the gold and silver.
8. Casting the silver and gold into ingots and bricks.

I. Receiving and Melting the Deposits.

The United States Mint receives upon deposit, bullion in bricks and bars from all parts of the United States, and, besides, parts all articles of gold or silver, which it receives on deposit, to be assayed and their value ascertained. Large quantities of old jewelry, and of new jewelry out of fashion, of all kinds, watch-cases, and any other materials containing gold and silver, are received. Anything containing a sufficient amount of the precious metals is received on deposit, except that, so far as is possible, material containing tin is rejected. This is the case with much of the cheap jewelry, which is filled with solder, and with watch-cases, which, within the past few years, have frequently been alloyed with tin, which is a difficult metal to separate with the appliances which are used at the mint. It might be melted with lead and cupelled, but there are no cupellation furnaces at the mint, and the refining by fluxes is very difficult, and gives a great deal of residue. Each deposit is melted by itself, and, if large enough, is cast in the slipper moulds, if not, into any kind of form. This deposit when melted, is numbered, assayed, and sent to the melting-room, where it is all mixed, so as to make the alloy of proper composition for parting, if it contains gold.

The deposit melting-room contains three furnaces of the same size as all the others in the mint. They use, however, charcoal as fuel. The gold is melted in Gunlack-Hessian crucibles, holding about 200 ounces each. The silver is melted in black-lead crucibles, Nos. 20 to 35, holding from 1000 to 1200 ounces each. The whole work of this department is done by three men and one superintendent. They use about 100 bushels of charcoal per week.

In melting the bullion a cover is almost always used. The crucible is so large that there is very little danger of the draft ever reaching the melted material. This cover is generally of charcoal or of bone-ash. When the material contains metals which are to be oxidized, this cover is made of bone-ash $\frac{1}{16}$ of an inch thick over the melted metal, and then nitre is added through holes made in it to oxidize the impurities.

According to the quality of the bullion, this cover is renewed from time to time as is required. If it is very impure it may be renewed three times. Usually it is not done more than once or twice. When the metal is sufficiently purified the cover is care-

fully removed and the metal fluxed with borax. The cover and the slag are ground, sifted, and washed to recover the small particles of bullion. All the material from the bullion deposit-room is toughened. If it is not sufficiently tough on the first melting, it is remelted with pulverized charcoal.

Many of the silver bars which come from New Mexico and other localities, contain considerable amounts of arsenic and antimony. This metal is refined by putting into the melted bullion an iron bar four feet long, three inches wide, and from one-half to one inch in thickness, with which the metal is briskly stirred. A little nitre is added to the charge, and the metal is dipped with an iron ladle. This will generally remove all the antimony after two or three minutes of stirring. Before this method was devised it was found extremely difficult to separate the antimony from the bullion. When tin is present, pearlash is used, and thoroughly mixed with metal. If the slag is too thick a little borax is added. When iron and sulphur are present in either gold or silver, a flux of one part sand and two of nitre is used. Bone-ash is used to thin the slag. When lead is present in either silver or gold, bone-ash is used. Nitre is added, and holes made through the cover from time to time as it seems best.

The fluxes which are generally used in melting are sand, borax, sal ammoniac, soda, salt, and nitre. Sand is generally used when the gold or silver bullion contains iron. Borax is used when the metals are quite pure; sal ammoniac when the metal contains tin, or other substances difficult to separate. Nitre is used as a general oxidizing reagent, especially when the metal contains lead. When the slags are thick, soda is used to thin them, so as to make them deposit the granules of metal, and, if this is not sufficient, salt is added to it. Whenever any of these fluxes are used the metal is not allowed to come within six or seven inches of the top of the crucible. When the metal is entirely fused it is left for some little time in quiet fusion, and then stirred with a graphite stirrer held in the tongs, so as to give the whole force of the upper part of the body in stirring to make the metal homogeneous, and not depend upon the wrists. Since this method has been introduced, very few melts have been found to be unequal in value at the top and the bottom of the ingot. As soon as it is stirred it is poured. When cast, the deposit is sent into the deposit-room to have its number

marked upon it, and, subsequently, to be assayed to determine its value and fineness.

They use in the deposit melting-room graphite crucibles, Nos. 12, 20, 35, 50, and 70, made by Taylor, of Philadelphia. These usually last a week, or about 100 melts. Of the Hessian crucibles, which come five in a nest, only the two largest ones are used. They last for five or six melts. Anthracite coal is used for large melts and charcoal for small ones.

When the deposits are received they are melted just as they come with fluxes and poured. If tough they are sent away at once. If not, they are melted under borax and charcoal until they are. Sometimes they require to be melted twice, and at others three times, until they are perfectly tough.

II. Inquartation and Granulation of the Material containing Gold.

The melting-furnaces are on the first story directly under the parting-room. Here all the gold and silver is melted either for granulation or for ingots, which are the strips to be rolled out to make coin. The room is exactly the same size as the parting-house, and is divided into two parts by a chimney, which has four furnaces on each side of it, and three at one end, making eleven in all, each of which takes a No. 70 graphite crucible. They melt 3500 ounces of standard silver and 6700 ounces of standard gold. The silver bars weigh from 20 to 180 ounces, and the gold from 100 to 200 ounces. The crucibles last 36 meltings of silver and 24 of gold. Ingot melting is done every day of the week. Granulation is done only once a week, the metal being made of three parts of silver and one of gold. The granulation is done in water, which is cooled with ice. It is done in a copper vessel with a pan like that used in the New York Assay Office,* with three handles caught by three hooks. The back hook is very short so as to throw the pan forward to one side which has holes at the bottom in the rim on the front side, so that the water can drain off. When the granulation is finished the water is drawn off with a copper siphon, which connects with a pipe in the floor leading to the sewer, and fresh water added. To

* School of Mines Quarterly, vol. vii., p. 9.

effect the granulation a wooden platform one foot high is placed in front of the furnace. The metal is dipped out of the crucible held high above the head, and poured in a thin stream with a slight wavy and rotary motion. The doré silver is not granulated, but is sent to the parting-house in ingots or bricks.

III. Solution of the Granulations in Nitric Acid.

The granulations are carried from the melting-room to the parting-room, which is on the floor above, in wooden boxes lined with copper, 18 by 13 inches and 6 inches deep, with two iron handles upon the outside for lifting them. The transportation is done on low platform wagons with iron wheels, which carry twelve or more boxes, and are raised to the level of the parting-house by a lift. The granulations are poured into a flaring copper pan 40 by 33 inches on the top and 24 by 30 inches across the bottom, and 8 inches deep. The rim is reinforced by a ¾-inch half-round copper bar. It has two handles to carry it by, and, if necessary, to lift it when full. It is raised by a Weston pulley-block. The pan is set upon a table 25 inches high, which is directly against the wall next to a pair of platform scales, on the same level as the table. The granules are taken out of it with a copper scoop and put into a copper receiver on the platform of the scales and weighed, so as to get the exact quantity in each pot. The room in which the parting is done is lighted on both sides and one end by large windows, but is not very light, owing to the proximity of other buildings. It is 80 feet long, 30 feet wide, and 12 feet 6 inches high, with a stone floor. The corroding-house is in the centre of one end. The pots are contained in a wooden house 23 feet long, 10 feet high, and 5 feet 6 inches wide, the bottom of which is raised 12 inches from the floor. The vertical sides of this house are 7 feet high. The roof is inclined; the peak of it is cut flat for a foot, and is two feet below the ceiling. The inside of this has a double lining of boards, which can be taken out when it is too much eaten. The whole house is painted on the inside with asphalt, and on the outside with ordinary paint. On the bottom it is divided into two parts, each of which has two vats. The larger of these houses receives six parting-pots on a side, and the other five. Each section of the house has two tanks lined with 12-pound lead, on the side of which, three inches from the wall, is a special plank-

used as a splash-board, to prevent the throwing up of the water by the steam when it is boiled. Each of these tanks is 2 feet wide, and the length of half the house. Its depth is 22 inches. They are made of 2-inch plank. On the bottom of each of them a pipe is placed for the circulation of the steam to heat up the water.

In each compartment of the house there are three doors on a side, which give access to two pots each, except the last door at the end of the smaller one, which gives access to only one. These doors are 40 inches wide and 45 inches high, and are made in wooden frames, the corners of which are reinforced by irons bent at right angles. They are counterpoised by weights hanging upon chains from each side, so as to allow the door to be opened to a height of 5 feet 9 inches above the ground, which is necessary to give the requisite space for handling the wooden stirrer used in the pots. Each one of these doors has two iron handles for the purpose of lifting it easily. The first house has all the doors alike; it is 12 feet 6 inches long. The other is 11 feet 6 inches long, and has two doors alike, and one of 28 inches, which gives access to only one pot. The whole house is braced with joists, which are 4 by 3 inches between each door. They rise a few inches higher than the bottom of the doors when they are opened.

All four of the tanks have water on the bottom to the depth of 10 inches. This depth is regulated by an overflow pipe, so that it is always the same. These overflow pipes are in the centre of the side of the house, and the steam for boiling comes in at the two ends.

The pots used are made of glazed earthenware. They come from Germany. Most of them are 21 inches in diameter and 22 inches deep. They vary a little, however, the limits being 21 to 24 inches in diameter and 21 to 22 inches in depth. Each of these has two conical handles on the outside, which are 6 inches wide and 5 inches high. Each has a spout 2 inches wide at the lip and 1½ inches deep. The weight of the corroding jars is 90 pounds. These are made in Germany, and do not crack by any heat that is produced by steam, and are only broken by accident. They were formerly made in this country, but caused so much loss by cracking, and were so easily broken, that only German ones are now used. They last for a long time, most of those on hand at the present time having been purchased during the war. They are set in the vats on a lattice-work of wood 1½ inches thick, to pro-

tect the bottom of the tank from injury, with holes 4 inches square. This rack is set between the steam-pipes which run around the sides. These pipes are of copper, 1½ inches in diameter, and make the round of the tank next to the four sides of it. They are pierced on the insides with ⅛-inch holes every 2 inches. The pipes are thus behind the splash-board, the steam being discharged towards the interior of the tank. A wooden paddle, 33 inches long, with the flat 3 inches wide, 1 inch thick and 9 inches long, is used in each one of these pots to stir the mixture. The flue for the whole house is at one end. It is 29 inches on the bottom and 1 foot high vertically. It then follows the slope of the roof so that at the highest point it is 2 feet. A flue from the chimney for cleaning the blanks for making the copper coins enters the side next the flue. From the brickwork the flue is continued in a terra-cotta pipe, 14 inches in diameter, and set in a brick flue bound with iron. It descends vertically 30 feet, and enters a horizontal covered culvert, 4 feet by 3, which is about 40 feet long, and connects with a chimney 8 feet 6 inches on the inside and 120 feet high. One hundred and twenty-five pounds of granulated bullion, a part of which, in some cases, consists of small silver coins which are condemned, are put into each one of these pots.

The granulated bullion accumulated during one month will fill from 22 to 28 pots. Five pitchers of nitric acid, which each hold 25 pounds of the acid at 40 B., are put into each pot, covering the bottom nearly 9 inches. The whole height of the charge in the pot, including granulations, will be 12 inches, which about half fills it. Steam is introduced into the pipes on the bottom of the tank and the water brought up to a boil and kept so for twelve hours. In order to prevent them from getting clotted the granulations must be stirred every twenty minutes, each stirring taking about a minute for each pot. No cover is placed over the pots. They are left uncovered for greater ease of stirring. The sliding door for this purpose is raised, but the draft is so strong that none of the fumes come out into the room. The only acid fumes come from the transfer of the various acids. The workmen leave the building at six o'clock, but the steam is kept up until 8 P.M., at which time it is shut off. During these two hours the house is looked after by the night watchman. The water to thin down the argentic nitrate is not added until the following morning, when the bath is heated up for a few minutes to dissolve any crystals

that may be formed. The water is added until it reaches to within 3 inches of the top of the corroding pot. The solution is then drawn off with a gold siphon 1½ inches in diameter; this has a taper wooden plug 2½ by 1¾ inches. The siphon costs $3600. The silver nitrate is collected in wooden tubs 18 inches in diameter and 15 inches high, made with staves 1¼ inches thick. They are held together by two hoops of 2-inch iron, and have handles on the inside for the purpose of carrying them. They last for two years. When the acid is drawn down to near the top of the gold three pitchers of fresh acid are added, and the granules are boiled for another twelve hours as before. Each charge thus remains in the pot forty-eight hours, during half of which it is constantly boiled. This last acid is not saturated. It is bailed into an adjoining empty pot in the house, a leaden strip being placed between the two, so as to prevent any spilling into the water-bath. This acid is used on a fresh charge.

The pitchers for the acid are of the same shape as the ordinary bedroom pitchers, but much larger. They are 18 inches high, 6 inches in diameter in the opening above, and 9 inches at the belly. Besides the usual handle on one side, they have also one on the belly to lift them with. They are made of ordinary enamelled earthenware. They weigh 10 pounds each when empty. The nitric acid is brought to the lift in carboys. These are carried on wagons which hold two each, and turned at once into earthen pots 18 inches in diameter and 28 high. From here it is drawn into the pitchers. There are four of these receivers. Five pitcherfuls are counted to a carboy. When the acid is bailed down so that there is danger of taking out the gold, water is added to dilute the rest of the acid, and it is siphoned into a wooden tub on the floor outside the house. When the acid is dilute enough, the pot is lifted out of the house and the gold washed out of it into a filter. This filter is a round wooden tank, made of staves 1¾ inches thick; these are reinforced on the outside with others 2¼ inches wide by 1 inch thick, leaving a space of 1½ inches between each stave. The whole is bound with three hoops of 2-inch iron ⅛ of an inch thick, and is painted on the outside. The filter is 28 inches in diameter on the inside at the top and 24 at the bottom. It has a false bottom, 4 inches above the permanent one, filled with ¼-inch holes, leaving 17 inches for the inner height of the filter. The false bottom is first covered with a layer of cotton cloth, which

is brought up over the top and tacked to it. One thickness of Swedish filter-paper, which takes three sheets, is put over the bottom. Thick stiff brown paper is then put round the sides. Then a thickness of muslin, which, like the paper, is carefully pleated on the sides, is tacked over the top of the tub. The filter-paper is thus between two thicknesses of muslin. This filter has a wooden cover made in three parts. The whole filter is set inside of an iron frame resting upon a wagon; the frame is so arranged that the four arms by which it is caught shut over the wooden cover and make it quite tight, and form two eyes at the top by which it is lifted. Between the false and the real bottom there is a large wooden stopcock, with a 1¾-inch hole projecting 12 inches from the side, and supported by two irons hooked to the inside of the filter. Into this filter the gold in the bottom of each vat is poured, and the pots thoroughly washed to take out the last traces of gold. It is then raised to the overhead crane and carried to the large silver tank, where it is suspended, and the gold then carefully washed with hot water, which enters from an iron pipe in the ceiling, and is allowed to run by itself until it is perfectly sweet. This takes from two to three hours.

The charge from 11 pots will be about one foot high on the bottom of the filter. When the gold is quite sweet the tank is lowered to the ground and the gold is removed from the cloth by earthenware dippers to be treated by sulphuric acid. The upper muslin of the filter lasts for two operations, the lower one lasts for three. When they are no longer strong enough to bear the weight of the gold, they are dried and burned, and the gold recovered added to the other gold.

IV. Boiling the Gold Residues Twice in Sulphuric Acid.

The gold from the filter is carried to 4 furnaces, which are hexagonal in shape, 25 inches high by 28 inches across. These are made of boiler iron with cast-iron doors and tops, and are lined with brick. Anthracite coal is used in them exclusively. The cast-iron cover is 1½ inches thick. The top has on the outside a rim ½-inch deep. In its centre is a hole 21 inches in diameter with an inclined flange 2 inches wide, having the same angle as the bottom of the sides of the pot in which the gold is to be treated. This rim rises half an inch above the top, so that there is a basin half an inch deep all around the top of the furnace to catch any-

thing that may be spilled over the sides of the cast-iron pots. This hole has a cover 1½ inches thick. In the top of this furnace a cast-iron pot, with three legs 3 inches long, is placed. It is nearly hemispherical, being 22 inches in diameter at the top and 12 inches deep in the centre. It has two trunnions. When set in the hole of the furnace 7½ inches of the pot are above it. The trunnions prevent its passing in any further. It is intended that the bottom of the pot should fit the flange pretty nearly tight. With these trunnions, when the operation is finished, the pot is lifted by two men, by placing a piece of gas-pipe over the trunnions, so as to give a handle long enough to get an easy purchase. The lid of the furnace is then slid under for the pot to rest on. It was originally intended to lift these pots by the trunnions with a Weston pulley-block and carry them away on an overhead railway, but it has been found easier to lift and carry them by these pipe handles. The pots are closely covered by a conical lead hood, with a pipe 6 inches in diameter made of 12-pound lead sliding in another 8 inches in diameter. The hood has a hole in the side 7 by 6 inches, which is covered with a piece of sheet lead bent over at the top, so that it can be easily removed when its contents are to be stirred. These hoods are counterpoised with weights so as to be easily lifted. Eight feet above the furnace is a reservoir 2 feet square, the length of the four furnaces, and lined with 12-pound lead. It is inclined to one corner. The 8-inch pipe enters the bottom and rises two inches above it. In the centre of the box is a flue leading to the terra-cotta pipe in the ventilating flue of the parting house. Whatever acid condenses in the box flows to the corner and is carried by a lead pipe to a vessel 7 inches in diameter and 12 inches high on the floor. The rest of the acid escapes into the flue.

The gold is dipped out of the corroding pots with a porcelain vessel 6 inches in diameter and 4 inches deep with a porcelain handle 5 inches long. Under this a porcelain vessel 10 inches in diameter and two inches deep is held to catch any gold that might possibly fall. Ten of the porcelain scoopfuls are put into each one of the iron pots. One pitcherful of sulphuric acid at 66 B. is then added in each pot. This acid weighs 32 pounds. It is boiled for 1½ hours. By this time it has nearly evaporated. A second pitcherful is put on and boiled for the same time. The pot must be stirred every ten or fifteen minutes, which is done through the

opening in front, with an iron rod the end of which is hammered out into a four-pronged rake, the prongs being 3 inches, and the whole rake 3 feet, long. This is done to prevent the gold from agglomerating on the bottom. By this time the gold is 996 to 998 fine. If wanted finer than this it is boiled a third time, and can easily be made 999½ fine. The excess of acid is poured off to be used again. The acid from the gold is put into a lead-lined vat 20 inches in diameter and 24 deep, lined with 8-pound lead, bound with three iron hoops and diluted with water, and the silver contained precipitated in the porcelain pot. The residue of sulphate of lead and silver is allowed to collect. The gold after being carefully washed with water is kept in a porcelain solution-pot under water. All the wash-water from the gold is treated in the same pot. When small samples are treated the gold residues are filtered on a frame two feet square the inside of which is 20 inches square, a single thickness of muslin is tacked to this and over the muslin two thicknesses of filter paper are placed. The gold is washed down the sides of the filter paper with a two-quart pitcher. The liquor is saved and added to the silver liquors. When sufficient gold has been collected it is carried to the drying furnace in the cellar, which is similar to the drying furnace for silver, but a little smaller. It is not pressed into cakes, but when dry is sent directly to the melting-room.

The sulphuric acid is stored in a vat made of 2½ inch wood, 23 by 19 by 20 inches deep. This is covered inside and out with 12-pound lead and holds 2½ carboys, which do not fill it quite to the top. The acid is dipped out of this with a two-quart pitcher. The large pitchers are set upon a lead plate 2 inches larger in diameter than the bottom of the pitcher to make sure of catching any acid that may drop from the outside. The tank and the pitchers are set upon a platform six feet long and an inch wider than the carboy, with a rim 2 inches deep, lined inside and out with 12-pound lead. The tank is set at one end, and the pitchers at the other. Each pitcher is set upon a lead basin so arranged as to allow the pitchers to drain perfectly from any acid that may be spilled on the outside.

V. Precipitation of the Argentic Nitrate with Salt.

All the nitrate solutions from the eleven pots go to a large tank to have the silver precipitated. This tank is ten feet by eight, and

four and a half deep. It is made of three-inch staves which are reinforced on the outside by others 3 inches wide by 1 inch thick, with the space of 3 inches between each stave. This is held by 6 hoops 2¼ inches in width of ⅛-inch iron. The tank is set 3 feet above the floor. It is 4 feet deep on the inside, and 4 feet 6 inches deep on the outside. The salt solution for precipitating the silver is made in a tank on the floor above, the water being allowed to take up as much salt as it will dissolve. The salt for this solution is brought up in bags which contain one bushel each. The solution is conveyed to the tub in iron pipes and is run into the tank until it is one foot deep on the bottom. The whole is stirred by a wooden dasher 2 feet wide by 1 foot high with a handle 3 feet longer than the longest diameter of the tub. It takes two hours to bring the solution from the corroding jars and thoroughly precipitate the silver from it. After 6 pots have been added the liquor is thoroughly stirred. A platform is placed on the outside across one of the longer diameters of the tank which is 6 feet high and 7 feet long, 18 inches wide in the centre and 34 at the sides, being cut out to fit the shape of the tub. The top of the platform is reached by 6 steps on wheels, 6 feet wide, which are rolled out of the way when the silver chloride is carried to the reducing tank. On the opposite side is a platform 28 inches by 32, hung to the sides by iron bars which can be moved to any part of the tank when required. It is 58 inches above the ground. A ladder is also used to reach any part of the tank when it is necessary. When the test shows that the silver is all out of the liquor it is again thoroughly stirred with a dasher and the heavy parts of the precipitate pushed to the side of the tank where the spout is on the long diameter of the tank. The whole of the liquor will be about 14 inches deep on the bottom of the tank. The spout by which it is emptied is 18 inches long, 4 inches in diameter, with a discharge hole 2¾ inches in diameter. It is supported by two iron rods attached to the side of the tank. This spout projects over a tank on a wagon the top of which is 2 feet above the bottom of the floor. This tank is 6 feet 6 inches long, by 2 feet 7 inches wide and 15 inches deep. It is lined with 12-pound lead. It is set upon a frame on the top of the wagon, which is 3 inches thick and 9 inches high. This top inclines to the centre and also to one end leaving a space of 3 inches at the lower end between the top and bottom of the tank. The bottom of the

tank is filled with holes ¼-inch in diameter, and over these and the sides of the tank twilled cloth is put and kept to its place on the sides with a frame made of half-inch stuff painted. Both the frame of the filter and the wagon top have their corners reinforced with iron. The silver chloride tank is independent. It rests upon a cleat on the large frame, which is securely fastened about 2 inches from the bottom of the frame. The top of the truck inclines toward the centre and one end. Under the centre of this end troughs made out of a single stick 7 inches wide and 4½ inches thick hollowed out in the middle are placed. The one next the tank projects a few inches under the bottom of the tank. The thickest of the silver chloride on the bottom of the precipitation tank is pushed by the dasher to the stop-cock and made to flow on to the filter till it forms a bed 2 or 3 inches in thickness. This is called a bottom. After this has been done the rest of the contents of the tank is made to flow over this bed, which is thick enough to prevent any fine silver chloride from passing through the cloth. While the silver chloride bottom is being formed the liquor does not run through the filter clear, and low wooden pails, 13 inches in diameter and 8 inches high, with copper hoops and handles, are set under the trough to catch it. As long as it is turbid this liquor is thrown back into the vat. When a glass full held against the light is perfectly clear, the pails are removed and a trough like the others, but 6 feet long, is placed under the bottom trough and carries the excess of liquor away. The silver chloride is washed for about four hours. It is tested both by color and by paper to see when it is perfectly sweet. The silver chloride is washed with cold water until sweet. This water runs through two or three traps and then goes to the sewer.

VI. Reduction of the Argentic Chloride with Zinc.

The wagon, with the silver chloride in the tank upon it, when it has been completely drained is moved to a tank between the precipitation tank and the gold-refining furnaces. This tank is 6 feet long by 3 feet 9 inches wide and 18 inches deep. It is lined with ¼-inch lead. This lead lining lasts for two or three years and then gives out in the seams. The corners of the bottom of this tank are rounded for greater ease in stirring. Four boxes 20 by 12 inches and 6

inches deep made of ½-inch wood with two iron handles on the outside, filled with granulated zinc, are thrown into the tank to reduce the silver chloride. These will generally be sufficient, but occasionally 4½ boxes will be required. The calculation is generally made that it will take 2 boxes for 11 solution pots, but it sometimes takes a little more. The zinc box weighs, when empty, 10 pounds, and when full of zinc 42 pounds, so that each box contains a charge of 32 pounds of zinc.

The silver chloride is shovelled out of the vat on wheels with a copper scoop shovel. The chloride is put into the tank first, and the zinc stirred through it afterwards. The action of reduction should commence at once; but if it does not, a pint of sulphuric acid is added to make it begin. It is stirred with a wooden stirrer, 4 feet 4 inches long, with the shovel part 10 by 17 inches, and 2½ inches thick, the end of which is cut down to a sharp edge. One man must stir it very actively for an hour. After this it need only be stirred occasionally. It takes four hours to finish the whole. When the reduced silver is very dry a little water is added to dissolve out the zinc sulphate.

This vat was formerly covered with a lead-lined hood counterpoised, and having a chimney, 14 inches square, going to the roof. This was raised from time to time as needed in the working. It was very heavy, and the men eventually became afraid of it, so it was taken down. They say they do not feel any great inconvenience from the gases evolved. The charge in the tank is 12 inches deep on the bottom. When there is no further action, three pitchers of sulphuric acid, weighing 32 pounds each, are added to dissolve out any excess of zinc. The vat is allowed to lie all night with the acid in it. The reduced silver is pushed to one side, and any liquor in excess is drawn off with a copper siphon.

The zinc is granulated in a copper tank like the bullion granulating tank. It is 32 inches in diameter at the top, and 27 inches deep, and slightly flaring. It has two handles on the side. A pan, 26 inches in diameter and 6 inches deep, fits into this. It has three handles, two at the ends of one diameter, and one at the diameter at right angles to it, and is in all respects like the silver granulating tank. The zinc is melted in a crucible in a furnace exactly like the melting-furnaces, and is poured into this tank with a rotary motion. A ¾ inch pipe discharges cold water into the tub; a 1¾-inch iron pipe, 15 feet long, carries the escaping water to the sewer.

When the pan is full it is raised by a pulley. The handles cause it to tip to one side. It is suspended over the top of the tank to drain, the front of the pan being perforated so that the water flows back into the vat. The zinc from the pan is thrown into a wooden, hopper-shaped reservoir on wheels. This is 35 by 30 inches on the top, and 40 inches high. On the bottom there is an opening one foot high the whole width of the box. In front the bottom extends into a shelf 16 inches wide. The granulated zinc is removed from here with a shovel. One of these cars full lasts for four reductions.

VII. Sweetening and Pressing the Reduced Silver into Cakes.

The reduced silver is taken out with a copper scoop 10 inches in diameter and 5 inches deep, with a shovel-handle, and put into a colander. This is a cylindrical, wooden vessel on wheels, lined with 8-lb. lead, which is turned over at the top. It is 3 feet in diameter, 20 inches deep to the false bottom, which is 6 inches above the other. It has four 1½-inch iron hoops. There is a one-inch overflow pipe at the top and running down the sides. The silver here is washed with hot water until it is entirely sweet.

Before putting in silver the lead bottom, which is filled with quarter-inch holes, is covered with one thickness of twilled cloth, and is filled to within four inches of the top. The water runs out as fast as it runs in. This water flows into a box with two partitions, and from there to a box with four partitions, and from there to the sewer. These boxes are simply used as a precaution. Not more than two or three ounces of silver per year is caught.

The nitre used in the melting was formerly made in the mint from the nitric acid used to brighten nickel coins. This was saturated with soda under a wooden hood, and concentrated at a very low heat in an iron vessel lined with lead, and crystallized for use in the melting department.

The articles contained in the parting-house are:

22 German corroding jars.
6 American " "
7 pitchers for nitric acid.
2 earthen reservoirs for nitric acid.

1 lead tank for sulphuric acid.
2 two-quart pitchers for "
7 wooden stirrers.
1 porcelain jar for the wooden stirrers.
1 wooden tub for copper scoop and shovel.
2 silver nitrate tubs.
2 salt trucks.
7 silver chloride dripping-tubs.
2 sulphuric tubs for cleaning gold.
2 large tub filters for gold.
1 " " " silver.
1 silver chloride tank.
2 " reduction tanks.
2 zinc vats on wheels.
4 boxes to measure charge of zinc.
6 carboy trucks for square boxes.
1 iron truck for round carboys.
1 copper scoop.
1 copper shovel.
1 porcelain scoop and basin.
4 iron stirring rakes for gold.
2 iron scrapers to clean up the floor.

Every time a colander is filled and washed, it is carried to the hydraulic press in the cellar. The silver is put into a circular mould 12 inches in diameter and 12 inches deep. The plunger of the hydraulic machine fits this mould rather loosely. The silver is patted down with a round wooden stamper 4 inches in diameter and 4 inches high, with a handle 5 inches long. The plunger is then made to descend until the cake is only 4 inches in thickness. About a gallon of water is forced out of each cake, which is caught on the filter. The silver is dried in the same kind of a furnace as the gold, but still contains water, so that it shrinks considerably in drying. The shrinkage amounts to nearly two inches in the diameter. The water from the hydraulic press formerly ran through six traps, but it is now caught on a twilled muslin filter; the water runs from that directly to the sewer.

In November, 1885, ten colanders were sent to the press. With six men three runs of eleven pots are the most ever done in a week.

The business of the mint is much less than it formerly was, for in 1848–9, during the gold excitement in California, 56 pots were done in a week.

The gold-drying furnace is 3 feet by 6, and 4 inches deep. It is made of ⅜ boiler iron, and is covered by a rectangular hood. From the top of this hood, in the middle, a pipe, 12 inches square, conducts the steam to the chimney which carries off the smoke. The front of the hood inclines a little, so that the top is slightly smaller than the bottom. It is 22 inches high. It is hinged so that it can be raised to give access to the whole of one side of the furnace. In the centre of this door there is a small one 11 inches square, which is used for observation during the process. An iron shovel and a copper scoop are used both to introduce and to remove the gold. It contains very little dust, as the gold from the sulphuric acid is generally coarse, and there is no fear of dust in melting, as the hood over the melting-pot stops the draft.

The silver-drying furnace is 7 feet by 38 inches, and 4 inches deep, the pan being made in the same way as the gold pan. It holds 20 cakes. The flue of the fireplace in both of these furnaces passes through the centre of each, and directly under the pan. The ash-pit is 2 feet deep. The wood used is pine, three feet long. There are no grates to the fireplace, but the end of the wood rests upon iron in front, and the brickwork at the back, which is protected from abrasion by an iron plate. The gold and the silver are brought in deep copper trays, set upon wagons. The trays for the gold are 39 inches by 20, and 16 inches deep. For the silver they are 36 x 24, and 18 inches deep. The silver cakes are piled up in the bottom, two in a row. There will thus be three rows, of six each, piled one on top of the other, and in the centre at the top two, care being taken, as far as possible, that they should not touch the copper sides. The cakes, as they come from the furnace, are shrunk unequally; those in the centre, directly over the flue, are the most contracted. They are often brought up to a red heat, and sometimes contract as much as two inches in the diameter. Those upon the sides frequently do not contract at all, or are very much less contracted than those in the centre.

The furnace is fired while some other work is being done. It requires very little attention. The fireplace is 14 inches square,

the wood being lighted with shavings put into the ash-pit and allowed to burn out, and another charge of wood put in. It usually takes three firings, with the furnace full of wood, to heat the silver, and 1½ for the gold.

VIII. Casting the Silver and Gold into Ingots and Bricks.

In the ingot melting-room the furnaces are arranged two upon one side against the central flue, with their coal-bins. One of the furnaces is used for granulation. There are, beside these, two others at the side of the room. In the next room there are three against the chimney and four at the end of the room on the main wall, making eleven in all. The crucibles used are of graphite. They are set in the furnace on half fire-bricks made for the purpose. When the ingots or bricks are too long for the pots graphite rings, 3 inches deep, are put over the top of the crucible with a cement of borax and silica. This melts and fastens the ring securely to the pot, making it perfectly tight. When the melt is finished, just before the last dip is taken from the crucible, this rim is pried up with a poker, and is easily removed. The pot is then used as before. These rings can be used three days.

When a pot is to be charged all the material is brought in a copper-lined box 18 by 13 and 6 inches deep. The part of the melt in small pieces is taken up first in a copper scoop. The larger pieces are then lifted upon the scoop by the hand. An iron hood, with two handles to lift it, is then placed over the pot, the round part of the bottom of which is the size of the outside of the top of the pot. The front fits over the incline of the front of the furnace, so as to effectually shield the men from the fire and prevent any danger of dropping a part of the charge into the fire. Usually there will be a little metal in the bottom of the pot from the previous melt, which cannot all be dipped out. The small pieces are added very carefully at first, and the larger ones as carefully on top of them to prevent any splashing. When the metal is in brick form, too large to go to the bottom of the pot, the cover is put over the brick on the door side, inclined so as to protect it as far as possible from the air. The fire is kept up as actively as possible. In order to prevent the air from coming in by the sides of the door, bars of iron are placed against it.

The time of melting is usually for silver one hour and ten minutes, for gold one hour and forty-five minutes, per melt. Anthracite coal is used exclusively in this department. When the metal is quite melted it is thoroughly stirred just before casting. For silver this is done with a round iron disk, 6 inches in diameter and ¼ inch thick, pierced with eleven rows of ¼-inch holes. This is set in a stirrup of ½-inch iron, the attachment being made to the side of the disk only. This stirrup is 9 inches long, and it is attached to the rod. The whole length inside the handle is 37 inches. The handle is bent round, and is 3 by 5 inches. It is moved up and down in the melted silver from the top to the bottom of the crucible. After some use this disk becomes bent, and the iron is eaten into, by the metal, in a very irregular manner, but always in the direction in which the iron has been rolled. These stirrers last, when in constant use, ten days. For gold, the stirrer is made of black lead. It is 17 inches long, 1¾ inches wide, and 1 inch thick on the upper part. The lower part gradually widens until, at the bottom, it is 3 inches wide. At this point there are three holes, two below and one above, ¼ inch in diameter, which go through the stirrer. The upper part is straight, but the lower part is slightly curved like a table-fork. The graphite stirrers for gold last three to four days. They are usually broken by accident, and do not wear out. They are held in a pair of pincers. The men step up on the wooden platform in front of the furnace and seize the tongs with both hands, stirring the metal in the pot with a movement from the shoulder, in this way stirring the metal most effectually with the least labor. The cups for dipping the gold and silver are also held in tongs when in use. They are made of graphite, expressly for this purpose, and last two or three days. They do not wear out, but are usually broken by accident. The pot covers last three or four days.

Connected with each fusion furnace are the following articles:

1 iron scoop for charcoal.

1 iron coal-bin and shovel.

1 iron scoop to hold the metal.

1 iron scoop, 10 by 6 inches and 3 inches deep, with handle 6 inches long, for collecting the scum taken from the pots.

1 iron shoe-mould to cast the metal in case of accident.

1 pan for granulated silver, 18 by 12 by 6 inches.
1 hood for charging the crucibles.
1 flat poker, 3 feet long by 1 inch wide and ¼ inch thick, turned up 1 inch at the end.
1 pair tongs to remove the grate bars.
1 spring tongs.
1 differential pulley for two furnaces.
1 duster.

Each furnace-man has a pair of crash mittens, made very heavy, and stuffed with old carpet, so as to be able to handle everything hot.

Two assays are made from each pot: the first after the first ingot has been poured, when some of the metal is granulated in a cup. The granulating cups are made of copper, and hold about a pint of water; twenty-four of them are set in a rack in two rows of two, and are filled with cold water. The hind one is marked with chalk No. 1, and the front one No. 2. The samples are marked to correspond. The number of the melt is also chalked upon the cup. The first melt is marked No. 1. The second is taken out just before the last ingot is poured, and is marked No. 2. If the assay shows that the metal is of a proper fineness, the ingots are sent to the coining department; if not, they are remelted. When the assay sample has been taken by chipping, the chips are put into a copper pan, which is marked in the same way.

Assay samples for standard silver are alone done by granulation. The assay samples for gold and fine silver are taken by chipping from the down corner. No doré metal ingots are made at the mint.

In thirteen hours six tons of standard silver ingots are cast, which keeps six furnaces occupied. The others are used either on gold or on granulations. The work is somewhat intermittent, but is quite severe while it lasts.

Including all the men of the deposit-room and parting-house, there are thirty-eight men employed in this department. In the melting department alone thirty-two men are constantly engaged. These men are occupied as follows:

8 opening and closing moulds.
7 melters.

4 fillers.
3 men in the parting-house.
3 men in the deposit melting-house.
3 grinding cinders, etc., in the cellar.
2 tapping ingots.
2 picking up and pickling ingots.
1 foreman in the deposit melting-room.
1 foreman in the coin melting-room.
1 overseer.
1 collier.
1 gatekeeper.
1 man at the hoist.

The loss in weight in the treatment is estimated at one-tenth ounce for every thousand ounces. The melting of standard silver is done under granulated charcoal, entirely free from dust, made for the purpose. On fine gold and silver the charcoal cover is only used to prevent the action of the air. Fine gold, which is base, is first mixed with fluxes, and charged in the pot when the hood is on it, and is melted very slowly. The fluxes used for separating the different materials from gold are as follows: When arsenic is present the gold is melted under a bone-ash cover, and a hole about 2 inches in diameter is made in the centre, nitre being poured in around the circumference after the metal is in quiet fusion. When this is saturated it is removed. It generally takes two or three hours to remove the arsenic, but it can be done in this way. When lead is present the same is done, but it takes less nitre, as the bone-ash absorbs most of the lead. When antimony is present the metal is liquated and the antimony separated in the same way as the arsenic, but every cup taken from the melt must be tested. It is best removed by cupellation with wood as a fuel, the flame striking the metal. When tin is present bone-ash and nitre are used. All the dampers in the flues are closed while working with the stirrer. The quantity of nitre added has to be very large.

For the impurities in silver, when antimony is present, the metal in fusion is stirred with a large piece of iron for about five minutes. When arsenic is present bone-ash and a little nitre are used. The cover is allowed to stand until the bone-ash flakes, when it is removed

and fresh bone-ash added until there is no flaking. For lead bone-ash is used alone. When tin is present in silver, if there is any gold, it concentrates in the gold, and is separated in the same way.

The whole floor of the melting-room is covered with an iron grating full of hexagonal holes. The grating is in hexagonal pieces, 12 inches across. They cost 8 cents a pound. This grating pays for itself every three months, when it is taken up and the floor swept, and the silver and gold which would otherwise be lost, recovered. Beside the coin that is made for circulation, silver bars 999½ fine, which vary from 20 to 180 ounces in weight, and gold bars, from 5 to 243 ounces in weight, with a fineness of 999, are made. These are stamped with their fineness, weight, and value, and pass current just as the larger bars and bricks made in New York.

The ingot moulds are made of cast iron, in three pieces, planed up so as to have two ingot moulds in the same piece. They are clamped together with a strong iron screw, so that they can be easily separated. All three of the parts are greased with lard, put on with a swab, before they are used. The lard is purchased, by contract, free from water. The kegs are stored in the basement of the building, and what is used is held in a copper box 12 inches by 6 and 4 inches deep, with two handles at each end. At one of the ends there is a space 4 inches long, and the depth of the box, to hold the grease. The other end is raised to within 2 inches of the top, and this grease, taken from the deep part of the box, is spread over the surface, and the grease taken up by the swab from here and spread over the mould. This is done so as not to get too much grease on the swab. Standing at right angles to each furnace is a table made of wood, whose top is 2 feet by 18 inches. It is 18 inches high, with two projecting handles at each end at the top for the convenience of carrying the table when it is necessary to move it. On the top of this is a cast-iron pan to receive the ingot moulds, which are set on the bottom in two rows. These pans are 4 by 2 feet on the top, and 3 feet by 14 inches on the bottom, and 6 inches deep. Their object is to catch any silver or gold that may be dropped from the cup during the casting. The sides flare to an angle of about 45°. This table, like all the other heavy tables, is set upon full hexagons of iron, the same size and thickness as the grating, so that when the floor is to be cleaned, these heavy tables need not be moved. The hexagons are 1 foot from side

to side. Eighteen double-ingot moulds are placed upon the table at one time. As soon as the mould is filled, it is removed by a man, wearing very heavy gloves, to a table in front. This table is 8 feet long, 2 feet wide, and 30 inches high. The top is made of wood 4 inches thick, covered with a plate of cast iron 2 inches thick, with four rims running the full length of the table, ¾ of an inch wide and 1 inch high, the object of which is to keep the hot moulds and the ingots off of the body of the cast iron, and thus prevent its becoming heated. One man brings the moulds and opens them on the same side of the table as the furnace. On the opposite side a man with a copper tongs seizes the two ingots from each mould and places them on the end of the table until three moulds are open. He then seizes the six ingots in the copper tongs and plunges them into a rack in a bath of weak acid. This is contained in a box 22 inches square and 16 inches deep, made of 2-inch wood, and lined with 18-pound lead. On the bottom of this box upright copper bars, set in a rectangular frame, are placed. This box is covered with a hood. A quart of commercial sulphuric acid is put into it. The hot ingots cool off in this pickle, the six together. When cool, they are removed by the same man and put into a box, 24 inches by 18 inches and 10 inches high, made of sheet copper, into which cold water runs in and out to wash off the sulphuric acid. The top of this box is 24 inches above the floor, and on the same level as the pickle box, and at a convenient distance below the table. It has the same kind of a rack as the pickle box into which the ingots are set. The small amount of silver in the pickle acid is precipitated, washed, and taken to the parting-house. After the ingots have remained in the water for a few minutes they are removed and put into a wooden box in front of the copper tank, and from there are collected to be prepared for the rolls. There are two of these cooling and pickling vats in the same room. As soon as removed these ingots are taken to a bench, where there are two men with iron vices, protected with copper plates over the jaws. The jaws are opened by a treadle commanded by the foot. The edges of the ingots are filed off roughly. There are four of these benches, with two men at each bench, two benches being in each division of the large room. As soon as filed they are collected, and go to a shear press to have the upper end of the ingots cut off, leaving the lower one wedge-

shaped as it came from the mould, to facilitate its entrance into the rolls. They are then weighed, and if the assay shows the metal of proper quality they are sent at once to the minting department.

The ingot moulds are made of various sizes for the different denominations of coin to be made. They are all double except the quarter eagle, which has four divisions. All these moulds are 13 inches in length, and are wedge-shaped at one end. Their width only is variable. For the $20-gold piece they are 1½ inches wide and ½ an inch thick; for the $10-gold piece they are 1⅛ inches wide; for the $5, $\frac{5}{16}$ of an inch; for the $3, $\frac{7}{16}$ of an inch; for the $2.50 $\frac{1}{10}$ of an inch, and four to a mould; for the $1, 1¼ inches. For the silver, the $1 is 1⅞ inches wide; the 50 cents, 1⅜ inches; the 25 cents, 1⅛ inches; for the 10 cents the dollar ingot is used, two rows being cut out of it.

The melting and pouring of the silver dollars takes 1⅔ hours, of the 50-cent pieces 1¾ hours, of the 25-cent pieces 2 hours. In these melts there are 74 dollar ingots, 85 fifty cent, 126 twenty-five cent, for the silver. For the gold the $20-pieces have 60 ingots to a melt, and take 1¼ hours; the $10-pieces have 96 to a melt, and take 1½ hours; the $5 and the $3-pieces 150 to a melt, and take 1⅔ hours; the $2.50-pieces 250 to a melt, and take 2 hours; the $1-pieces 96 to a melt, and take 1½ hours.

The different denominations of coin have each its own special benches. Each of these benches has eighteen moulds. For the silver coins, there are six benches for the dollar ingots, three for the half-dollar, three for the quarter-dollar, and for the ten-cent the same as the dollar. All the ten-cent pieces are made at the dollar tables. For gold, there are two for double eagles, two for eagles, two for half-eagles, eighteen for quarter-eagles, ten for three-dollar pieces, and thirteen for one-dollar pieces.

List of articles used in the melting-room:

36 wooden boxes lined with copper for alloy.
6 trucks for carrying alloy, etc.
36 alloy boxes, 17 by 11 by 4½ inches, for the silver melts.
72 ingot boxes 18 by 12 by 6 inches.
60 granulating boxes 20 by 15 by 5 inches.

2 boxes for silver toppings.
1 box for gold toppings.
300 No. 70 graphite crucibles for silver and gold.
250 dipping cups.
250 No. 70 rings.
250 No. 70 pot covers.

The following table gives the legal weight and fineness of the coins of the United States, and their diameter and thickness:

	Legal weight (grains).	Fineness (thousandths).	Diameter (20ths of an inch).	Thickness (100ths of an inch).
Gold:				
Double eagle	516	900	27	77
Eagle	258	900	21	60
Half eagle	129	900	17	46
Three dollars	77.4	900	16	34
Quarter eagle	64.5	900	15	34
Dollar (new)	25.8	900	11	18
Silver:				
Trade dollar	420	900	30	82
Standard dollar	412.5	900	30	80
Half dollar	192.9	900	24	57
Quarter dollar	96.45	900	19	45
Twenty cents	77.16	900	17½	47
Dime	38.58	900	14	32
Half dime	19.2	900	12	23
Three cents	11.52	900	11	18
Minor:				
Five cents	77.16	75% copper, 25% nickel	16	62
Three cents	30	75% copper, 25% nickel	14½	34
Two cents	96	95% copper, 5% tin and zinc	18	60
One cent	48	95% copper, 5% tin and zinc	15	43

The table below gives the organization of the mint at Philadelphia.

GENERAL OFFICERS.

1 Superintendent.	1 Assistant assayer.	1 Register of deposits.
1 Assayer.	1 Ass't melter and refiner.	1 Assistant weigh clerk.
1 Melter and refiner.	1 Bookkeeper.	1 Assayer's computing clerk
1 Coiner.	1 Weigh clerk.	1 Medal clerk.
1 Engraver.	1 Abstract clerk.	1 Assistant bookkeeper.
1 Cashier.	1 Cashier's clerk.	1 Register of warrants.
1 Chief Clerk.	1 Warrant clerk.	

ASSISTANTS, WORKMEN AND ADJUSTERS.

General Department.	*Assayer's Department.*	*Coiner's Department.*
1 Doorkeeper.	1 Assistant assayer.	1 Chief weigher.
1 Messenger.	1 Weigher.	1 Assistant weigher.
1 Painter.	1 Assistant.	1 Hoister.
1 Gas fitter.	1 Foreman.	1 General foreman.
1 Gatekeeper.	1 Helper.	7 Counters (3 ladies).
1 Helper.	2 Workmen.	29 Coining room employees (21 ladies).
6 Assistants.	*Melter and Refiner's Department.*	10 Annealers.
5 Conductors.	1 Office clerk.	9 Cleaners.
6 Cabinet employés.	1 Foreman.	12 Rollers.
16 Night watchmen.	12 Melters.	16 Cutters.
6 Carpenters.	4 Deposit melters.	3 Medal room employés.
2 Engineers.	1 Refiner.	83 Adjusters.
1 Fireman.	2 Sweep cellar operatives.	51 Temporary adjusters.
1 Oiler.	1 Furnace builder.	*Engraver's Department.*
8 Machinists.	1 Hoister.	2 Engravers.
2 Adjusters of scales.	1 Gatekeeper.	5 Die makers.
3 Blacksmiths.	1 Laborer.	1 Assistant.
28 Laborers.	19 Helpers.	
12 Female helpers.		

The chimneys are cleaned once every year, during the last week in June, previous to making up the final settlement. The front of the flue to the ceiling is first taken down and carefully scraped. Most of the flue dust containing gold is to be found here. The surfaces of these bricks are very carefully scraped. The doors and windows all being closed, a steel brush is run up and down the rest of the flue from the roof, the top of the flue being covered while this operation is being performed. All the material so collected is put through the Kent's amalgamator.

The following table gives the wastage and losses on sales of these sweeps for 1885.* These losses are paid from surplus bul-

* Report of the Director of the United States Mint for 1885.

lion, as far as practicable, and the balance reimbursed from the appropriations and silver profit fund:

LOSSES.

Melter and refiner's gold wastage,	$146 70
Coiner's gold wastage,	50 81
Melter and refiner's silver wastage,	1561 81
Coiner's silver wastage,	3089 26
Loss on sale of sweeps,	4593 04
Total,	$9441 62

EXPENDITURES.

From contingent appropriation,	$483 69
From parting and refining appropriation,	457 24
From surplus bullion,	1491 24
From silver profit fund,	7009 45
Total,	$9441 62

The following are the charges imposed at the Coinage Mints and Assay Office at New York on deposits and purchases of bullion.

Deposit melting charge* on all bullion, or coin not required to be parted or refined:

For each melt of 1000 ounces or less,	100 cents.
Over 1000 ounces,	$\frac{1}{10}$ of 1 cent per oz.

Parting and refining charges. I.—*Parting Gold and Silver, or Refining Gold.*

(Rate per ounce, gross.)

On bullion containing	At Philadelphia.	At New Orleans.	At New York.	At San Francisco.	At Carson.
Under 200 M, gold	1½ cents.	1½ cents.	1 cent.	2 cents.	2 cents.
From 200 M to 399½ M, gold	2 "	2 "	2 cents.	3 "	3 "
From 400 M to 699½ M, gold	3 "	3 "	3 "	4 "	4 "
From 700 M and over, gold	4 "	4 "	4 "	6 "	6 "
Over 100 M, base metal additional	1 cent.	1 cent.	1 cent.	½ cent.	½ cent.

In addition to the above charges, on deposits requiring parting, or refining, for each deposit of 1000 ounces or less, a charge of 100 cents; over 1000 ounces, $\frac{1}{10}$ of 1 cent per ounce, gross is made. For gold coin or standard gold bars, the rate per ounce charged

* Report of the Director of the United States Mint for 1885, p. 133.

is imposed only on the number of ounces required to be refined to raise the whole to standard.

Silver allowed the depositor is calculated on the basis of refining the gold to 990 M.

II.—*Refining Silver.*

(Rate per ounce, gross.)

On bullion containing	At Philadelphia.	At New Orleans.	At New York.	At San Francisco.	At Carson.
Less than 897 M, silver.....	1½ cents.	1½ cents.	1½ cents.	2 cents.	2 cents.
From 897 M to 979½ M, silver...	1 cent.	1 cent.	1 cent.	1½ "	1½ "
From 980 M to 997½ M, silver...	½ "	½ "	½ "	1 cent.	1 cent.

In addition to the above, on silver deposits requiring refining, a charge on each deposit of 1000 ounces or less, 100 cents; over 1000 ounces, $\frac{1}{10}$ of 1 cent per ounce, gross is made. For standard or sterling bars the rate per ounce is charged only on the number of ounces required to be refined to raise the whole to the fineness of such bars.

TOUGHENING CHARGE.

Gold bullion, ... ½ to 2 cents per ounce, gross.
Silver bullion, ... ¼ to 1 cent per ounce, gross.

ALLOY CHARGE.

On the number of ounces of copper required to reduce the bullion to standard, ... 2 cents per ounce, troy.

BAR CHARGE.

On bullion deposited for fine bars not required to be parted or refined, and for standard, sterling, or unparted bars:

Bars of	At Philadelphia.	At New Orleans.	At New York.	At San Francisco.	At Carson.
Fine gold, per $100 value.....	5 cents.	5 cents.	4 cents.	10 cents.	10 cents.
Fine silver, per ounce, fine.....	¼ cent.	¼ cent.	$\frac{1}{10}$ cent.	¼ cent.	¼ cent.
Standard gold, per $100 value.....	10 cents.	10 cents.	10 cents.	10 cents.	10 cents.
Standard silver, per ounce, standard	½ cent.	½ cent.	½ cent.	½ cent.	½ cent.
Sterling silver, per ounce, gross....	⅝ "	⅝ "	⅝ "	⅝ "	⅝ "
Unparted bullion, per ounce, gross	½ "	½ "	½ "	½ "	½ "

Charges imposed at the Assay Offices, other than New York, on deposits of bullion, at the Mint at Denver conducted as an Assay Office, and the Assay Offices at Charlotte, N. C.; Boise, Idaho; Helena, Mont.; and St. Louis, Mo.

The operations of these institutions are confined by law to the receipt, melting, assaying, and stamping of bullion, and returning to depositors the identical bullion in unparted bars, or, in the case of gold bullion, the value in coin or unparted bars, as may be preferred by the depositor.

1. For melting, refining by fluxes, assaying and stamping of bullion, ⅛ of 1 per cent. on the gross value of the bullion.

2. When more than one melt is necessary, or extra materials are required to remove the base metals contained in deposits to render the resulting bullion ductile, an extra charge corresponding therewith must be made.

3. Upon gold bullion deposited for payment of its value from the bullion fund, in addition to the assay office charge, the same charges as those imposed by the mint to which such bullion is required to be forwarded, and a further charge to cover the cost of transportation, will be collected of the depositor.

The following table gives the earnings and expenditures of the mint for the year 1884–85.*

EARNINGS.

Parting and refining charges,	$12,188 02
Melting, alloy, and bar charges,	633 36
Charge for assaying, melting, and stamping,	....
Seignorage on standard silver dollars coined,	2,250,873 35
Seignorage on subsidiary silver coined,	10,197 61
Seignorage on minor coins manufactured,	452,351 40
Profits on sale of medals and proof coins,	2,566 59
Receipts from assays of ores and bullion,	392 00
Deposit melting-room fluxes and sweeps,	1,257 73
Surplus bullion returned by melter and refiners,	....
Gain on bullion shipped the mint for coinage,	....
Proceeds of sale of old material,	712 31
Profits from redemption fund,	12 99
Total,	$2,731,185 36

* Report of the Director of the United States Mint for 1885.

EXPENDITURES AND LOSSES.

Salaries of officers and clerks,	$40,673 91
Wages of workmen and adjusters,	367,854 51
Contingent expenses, less amount paid for wastage and loss on sweeps sold,	127,259 82
Parting and refining expenses, less amount paid for wastage and loss on sweeps sold,	7,793 73
Wastages of the operative departments,	4,848 58
Loss on sweeps of the year,	4,593 04
Expenses of distributing silver coins,	28,922 34
Expenses of distributing minor coins,	12,251 98
Minor coinage metal wasted,	405 29
Loss on sale of plumbic melts,	....
Total,	$594,603 20

In conclusion I wish to express my thanks to the officers of the mint who have given me every facility for studying this very interesting process.

OFFICERS AND MEMBERS OF THE ALUMNI ASSOCIATION OF THE SCHOOL OF MINES.

President: Frederick Augustus Schermerhorn, E.M., '68.
First Vice-President: Malvern Wells Iles, Ph.D., '75.
Second Vice-President: Frederick Remsen Hutton, Ph.D., '76.
Third Vice-President: Walter Bourchier Devereux, E.M., '78.
Treasurer: John Krom Rees, E.M., '75.
Secretary: Pierre de Peyster Ricketts, Ph.D., '71.
Assistant Secretary: Thomas Beale Stearns, E.M., '81.

Managers:

'67. John Adams Church, Ph.D.
'68. Frederick Augustus Schermerhorn, E.M.
'69. John Cooper F. Randolph, E.M.
'70. Elwyn Waller, Ph.D.
'71. Pierre de Peyster Ricketts, Ph.D.
'72. Peter Townsend Austen, Ph.D.
'73. Charles Adams Colton, E.M.
'74. Francis Bell Forsyth Rhodes, E.M.
'75. John Krom Rees, E.M.
'76. Frederick Remsen Hutton, Ph.D.
'77. Walter Edwards Hildreth, E.M.
'78. Willard Parker Butler, E.M.
'79. Nathaniel Lord Britton, Ph.D.
'80. Alfred Lockwood Beebe, Ph.B.
'81. Alvan Howard Van Sinderen, Ph.B.
'82. Alfred Joseph Moses, E.M.
'83. John Henry Banks, E.M.
'84. James Furman Kemp, E.M.
'85. Leon Marié, E.M.

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William R. Ware, B.S., Professor of Architecture, School of Mines, Columbia College. Address, 448 Madison Avenue, New York City.

REGULAR AND ASSOCIATE MEMBERS.

First Graduate of the School of Mines: John Magnus Adams, A.B., A.M., E.M., Mining and Mechanical Engineer, Agent for Frue Vanner, etc., 109 California Street, Room 7, San Francisco, Cal.

1867.

Francis Gordon Brown, E.M., Merchant, 116 East Twenty-seventh Street, New York City.

Edward Steele Bronson, A.B., A.M., E.M. Residence, 39 Garden Place, Brooklyn, N. Y.

John Adams Church, E.M., Ph.D., Tarrytown, N. Y.

Henry Bedinger Cornwall, A.B., A.M., E.M., Professor of Analytical Chemistry and Mineralogy, College of New Jersey, Princeton, N. J.

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William Wey Tuttle, E.M., North Springfield, Mo.

1868.

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George Strong Baxter, A.B., E.M., Mining and Civil Engineer, Northern Pacific Railroad, St. Paul, Minn.

James Petigru Carson, E.M. Address, care of Geo. E. Harding, 40 Exchange Place, New York City.

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G. M. Ogden, Trustee of Columbia College, Died July 18th, 1884.

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1868. Archibald MacMartin, A.B., A.M., E.M., Died in New York City, May 7th, 1881.
1869. Thomas Monahan Blossom, A.B., A.M., E.M., Died in California, October, 1876.
1869. Henry Newton, A.B., E.M., Ph.D., Died in Deadwood City, Dakota Territory, August 5th, 1877.
1872. Frank B. Jenney, E.M., Died in Trinidad, 1876.
1875. Robert Schuyler Lamson, C.E., Died in Darbour, Upper Egypt, in 1876.
1876. Frederick Everett Bruen, E.M., C.E., Died in Leadville, Colorado, November 8th, 1884.
1877. Henry Gilbert Clark, E.M., C.E., Died in Aurora, Nevada, August 2d, 1881.
1877. Frank Stuart Helleburg, C.E., Died in Leadville, Colorado, October 4th, 1883.
1877. James Robert Priest, E.M., C.E., Died in Greenville, Liberia, Africa, in 1880.
1878. Anton Fernekes, E.M., Died in Milwaukee, Wisconsin, March 13th, 1884.
1878. Henry Albert Hodges, E.M., Died in New York City, July 4th, 1883.
1878. Owen Frederick Olmstead, C.E., Died in Albany, New York, November 21st, 1881.
1879. Francis Barretto Stewart, Ph.B., Died in Harlem, New York, August, 1879.
1880. Frederick Denison Browning, E.M., Died in Colorado Springs, Colorado, August 25th, 1885.

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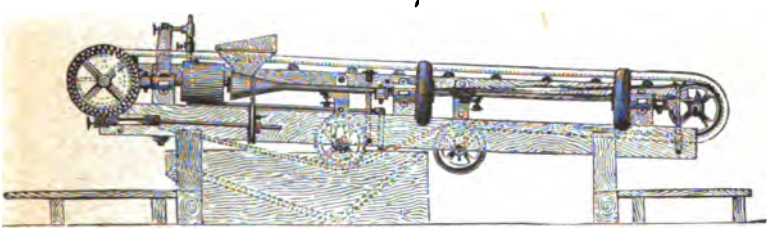

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1886.

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THE PYRITES DEPOSITS OF THE ALLEGHANIES.

BY ARTHUR F. WENDT.

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The Vershire Mines.

SEVEN miles west of the little town of Corinth, on the Passumpsic Railroad, in Vermont, are found the "Vershire" copper mines. The road from Corinth climbs a hilly country. Bold bluffs of granite and gneiss bar the way. Approaching the mines, we meet the usual talcose slates. Halfway up, on a steep hillside, the pyrites were first discovered, dipping at a low angle into the hill. On their strike they were traced several hundreds of feet, but only a short distance were they deemed worthy of the exploitation, which has followed them into great depths—over 1800 feet. Entrance to the mine is had through an adit about 400 feet in length. Where it intersects the old excavations, now void of ore, a boiler and hoisting-engine are located.

From here an incline, at an angle of 26°, follows the footwall to a bend in the track some 700 feet from the top; at this point the incline branches, and three diverging tracks join as one; they extend to the bottom of the workings. A large unbalanced iron car, holding two tons of ore or rock, can be switched on to either of the three diverging inclines, and this alone serves to raise the product of the mine. The car makes fifty trips per day. Following the abandoned excavations of the Vershire mine, from the

surface down, a clear picture of the occurrence of these ore beds is obtained. Three times in the history of the mine the end of the ore seemed to have been reached. The hanging and foot closed in, until the ore disappeared. As a matter of fact, the limits of that lens or lenticular mass had been reached, and it was exhausted. But invariably, on sinking and crosscutting into the foot, the edge of a new deposit rewarded the work of the explorer. At the date of the writer's examination of the mine, a face of ore twenty feet thick in the middle, and nearly one hundred feet long, was worked in the bottom. On both sides the ore pinched; on one side gradually until a mere seam was left, on the other it assumed a forked shape as if a wedge of country rock had been forced into the body of the ore. Underhand stoping was carried on and the ore hoisted as rapidly as it was broken. As a result, no development in advance of the mining could be done, and if ever the end of the lens of ore was reached, the production ceased and operations were at a standstill until the discovery of a new ore body. Such reckless mining brought its own reward in the difficulties the mine has since encountered. The character of the mineral of this mine is that of the "slate" ore beds, that is silicious. The percentage of copper is unusually high, the produce for many years averaging over 6 per cent. To the extent of the increased percentage of iron, due to the larger amount of chalcopyrite, is the ore more basic than other slate ores.

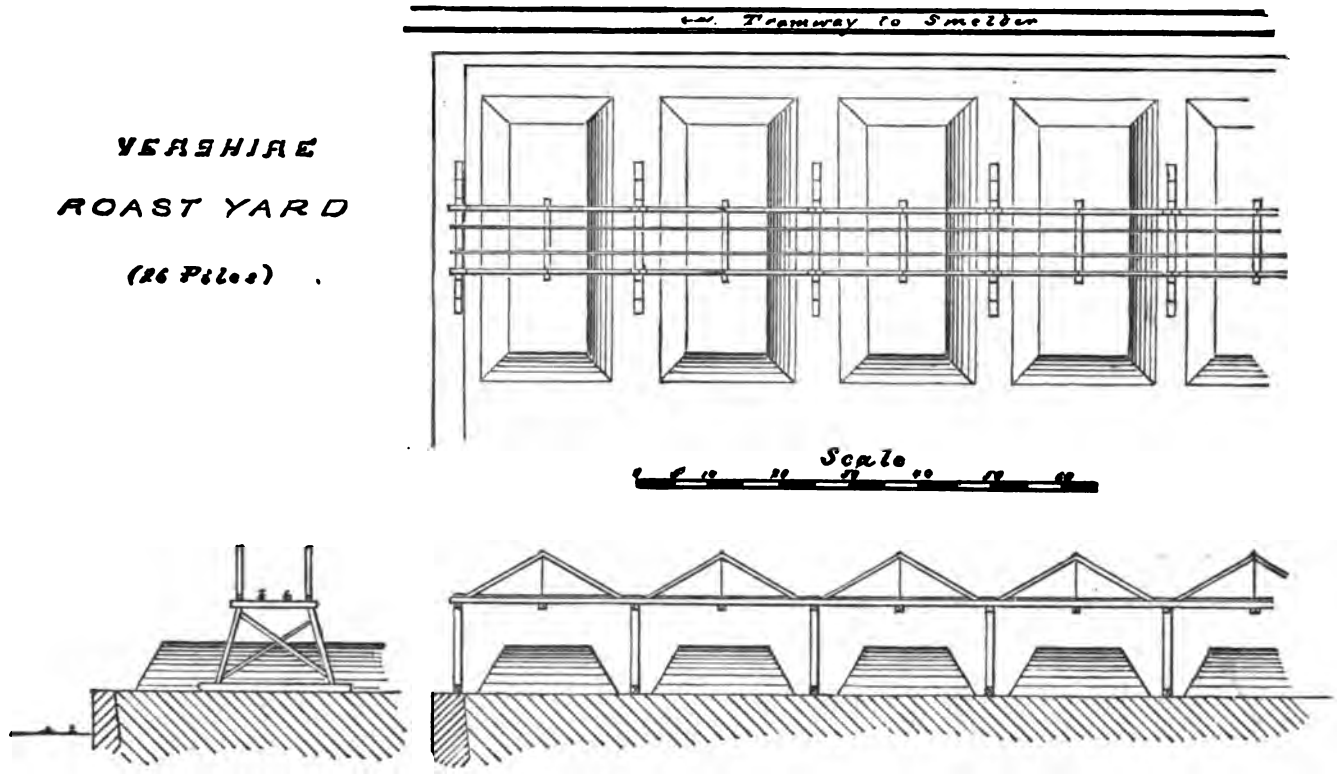
At the mouth of the adit tunnel the mine-cars are dumped on the cobbing floors. Rock, mundic, and low grade ore in large lumps are thrown over the dump. Of the remainder, the rich, coarse ore was cobbed and cracked, and sent to the roast-yard. The fine ore was thrown on hand-riddles, and the screenings were accumulated in a pile and allowed to agglomerate by slow oxidation. The ore remaining on the sieves was carefully handpicked by boys.

An inclined plane connected the cobbing floor with the roast-yard. The loaded cars drew up the empty. Of the arrangement of the roast-yard the accompanying drawing gives details.

The average of the ore sent to the roast-yard contained as follows:

Silica,	15 per cent.
Sulphur,	25 per cent.
Copper,	7 per cent.

FIG. 19.

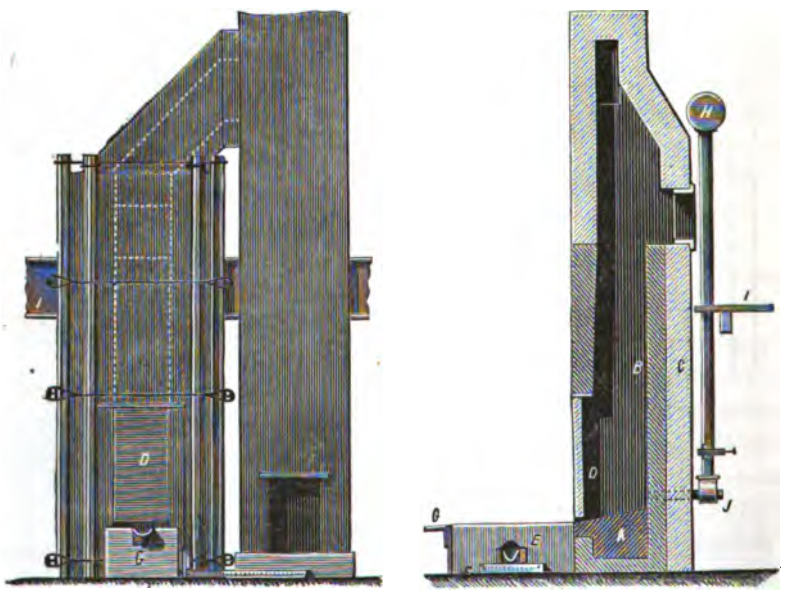


The manner of roasting was as follows (see cut): On a hard bottom, measuring 25 x 40 feet (much of it of rock), a layer of fine ore, 10 inches in thickness, was evenly spread. Two layers of cord-wood were laid on this, and on the wood the ore was dumped from the overhead track and levelled off to a uniform height of seven feet. Each pile, thus arranged, held 400 net tons of ore and was covered on both top and sides with a thick layer of fine ore, well beaten down.

During a roasting of six months, until the pile became extinct, the access of air, through cracks in the fine-ore cover, was carefully guarded. As a result of such careful work the ore, roasted in only one fire, assayed uniformly as low as 5 per cent. of sulphur. After cooling, the cover was stripped off, and together with the fine ore bottom was broken to fist size and finished in rock roast-stalls 12 feet wide, 8 feet deep, and 8 feet high.

The roasted ore was run by tram-road to the ore-furnaces, and was there smelted without flux other than an occasional dose of sand.

FIG. 20. FIG. 21.



Vershire Ore Furnaces.

Ore Smelting.—The smelting of the roasted ore was carried on in shaft-furnaces, of which the woodcuts give details. Roasted ore from the roast-yard arrived by the tram-road behind the furnaces and was dumped on the charge-floor, I. Coke, foul slag,

and dust from the stack were kept in separate piles, behind each furnace. No charges were made up, but the relative weights of stock and fuel charged were gauged entirely by the shovel; eight scoops of gas-house coke carried a burden of thirty shovels of the ore, slag, etc., equivalent to a burden of six pounds ore per pound of fuel.

Each furnace smelted 25,000 pounds per day. The attendance consisted of a furnace-man and helper, or slag-roller, at the bottom, and half of one charger at the top. By reason of the proximity of the furnaces it became possible for one charger to attend two furnaces.

Referring to the drawing, the absence of a crucible in the furnace proper will be noted. The smelted matte and slag is not allowed to gather in a crucible inside of the furnace, but runs out as fast as molten and collects in the crucible cut in a fore-hearth, E, of tamped steep, held in place by cast-iron plates. Slag runs continuously over the slag-lip G on the front of this fore-hearth, while the matte settles to the bottom in the exterior crucible, and is tapped at intervals through a tapping-notch, E, on a cast-iron plate, F, brushed with clay-water. The exterior crucible is remade at the end of every shift by packing with fresh steep. The stream of melted matte and slag leaving the furnace is changed from one side to another whenever it has cut an appreciable furrow in the surface of the steep fore-hearth.

A is a bottom of quartz; C the red brick body of the furnace; B the shaft of fire brick; D the breast only 4½ inches thick; H the blast main; I charge floor; J two tuyeres.

It is by this simple system of at once removing the melted material from the furnace proper that the troublesome sows are avoided.

Practically the same principle, although in a more modern form, is used in the Herresshoff and Orford furnaces, still to be described, and has been used in Germany for many years. Such furnaces are there called spectacle furnaces.

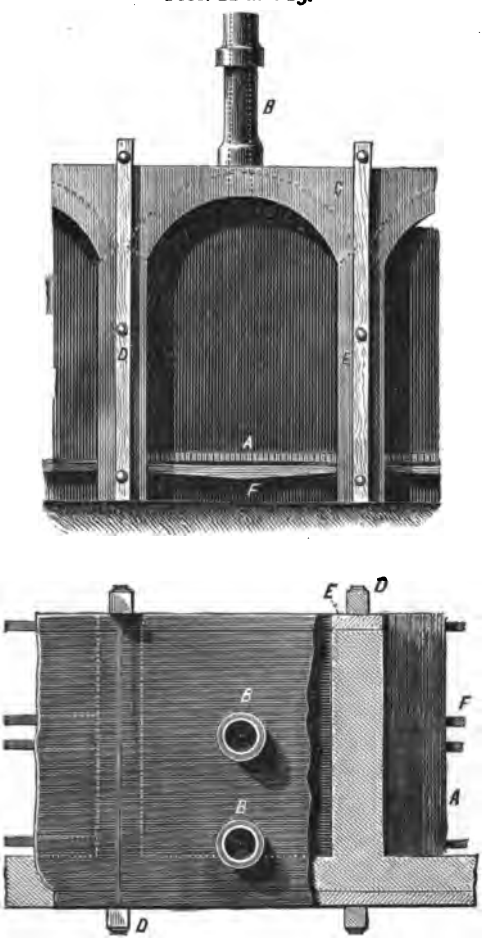
As stated the bottoms of the Vershire furnaces were packed with crushed quartz, the lining built of fire-brick. At the end of two weeks both need repairs. New quartz is packed on the bottom, and the repairs of the lining made with an average of only 250 bricks. The whole stoppage for repairs takes scarcely a day, as against a week or more in the Ducktown and Ore Knob practice.

The slags produced are of the same character and have the same composition as those made at the last-named places, but the matte

never falls below 30 per cent. of copper. This is partly due to the richer ore, but mainly to the careful and complete roasting.

A pair of Sturtevant blowers run by a turbine, or, in the event of a failure of water-power, by an engine, furnish the blast for all

FIGS. 22 and 23.



Vershire Matte Roasting Stalls.

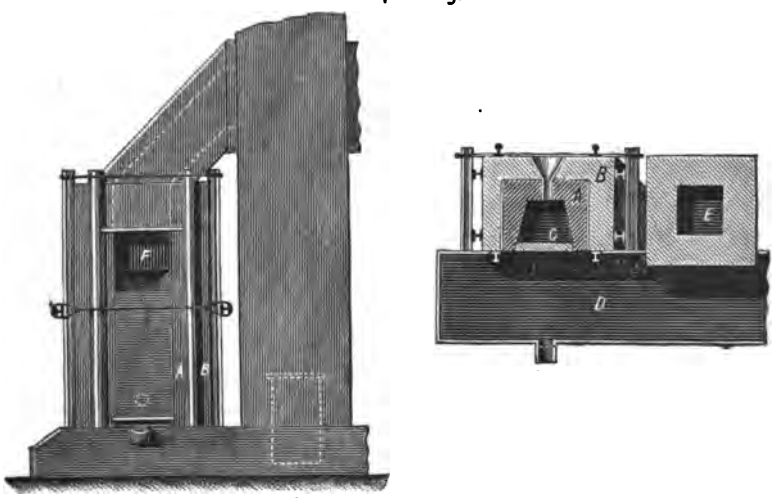
the furnaces. They are arranged in one line, and form a very compact plant, both less expensive in first cost and better arranged for economy of labor than any described so far.

Matte Roasting at Vershire was carried out in stalls, and with a care equal to that of the ore-roasting. A drawing of the stalls ex-

plains their construction. Two rows of these stalls were built facing one another, with a paved floor between. The first matte cast in sheets on the iron plates is cracked, and piled to a height of 8 feet on the iron grate-bars, A, of the stalls. For a height of 3 or 4 feet the front of the stall is closed with loose brick, and the open front above the brick temporarily closed with an iron sheet while firing the matte. Brushwood burned below the grate-bars serves to ignite the matte. The sheet-iron is then removed and the matte allowed to burn with practically no draft. The two chimneys, made of drain-tiles, B, carry off the smoke and make the work between the two rows of stalls comparatively pleasant. D are wooden buck-staves; F supports for the grate-bars; C and E the red-brick body of the stalls.

Four fires are necessary to reduce the sulphur sufficiently to admit of black copper smelting. Second matte from that smelting was mixed through the first matte and again roasted.

FIGS. 24 and 25.



Vershire Black Copper Furnace.

Black Copper Smelting.—The cuts of the black copper furnaces show them to have been both small and crude in design. The smelting process at Vershire, up to this point, was decidedly superior to that of the other works. Here superiority ceases, and in its place a retrograde movement sets in. We find the principle of a detached exterior crucible carried out in black copper smelting

without commensurate advantage. Black copper furnaces do not "iron" in the crucible, hence nothing is gained there. But they always tend to produce slags rich in copper, unless the crucible is large and hot, and thus admits of the gradual settling of particles of black copper, suspended in the molten slag. C is the shaft of the furnace with one tuyere; A the fire-brick lining; B the shell of the furnace; E the stack; D the fore-hearth of steep, common to a number of these furnaces.

The small matte furnaces smelted only 12,000 pounds stock per diem. They are charged from in front by the furnace-man, who places the charge in a small box, carries it to the front, and standing on the fore-hearth lifts it bodily and throws it on top of the blazing coke previously introduced in like manner. Two crucibles are cut into the steep of the fore-hearth, and serve alternately to arrest the black copper and rich matte. When one crucible is filled the molten current is guided into the empty crucible. The slag covering the full crucible is skimmed until the second matte is reached below. This is removed in thin disks or crusts with a pronged iron tool, and finally the black copper is ladled into moulds to be sold as such. It contains from 90 to 95 per cent. of fine copper.

A large proportion of the slag made in these small matte-furnaces contains over 1 per cent. of copper and must be returned to the ore-furnaces. Skulls of black copper forming in the small exterior crucibles are added to the charge of the matte furnaces.

The following were the rates of wages and cost of materials at Vershire in 1877 and 1878:

Gas coke at smelter,	½ cent per pound.
Wood,	$4 per cord.
No. 1 firebrick,	$100 per M.
Freight to and from R.R.,	$2.75 per ton.
Miners,	$1.50 per day.
Common labor,	$1 per day.

At this comparatively low cost of fuel, labor, and supplies, the total cost of smelting, with the plant described, must have been quite low, but little, if any, above five dollars per net ton of ore treated. The ore yielded considerably in excess of any of the pyrites mines described heretofore, and the works should have netted a handsome margin in the years gone by, when copper

averaged from fifteen to twenty-five cents per pound. That this has not been the case, is attributable to the system of mining before referred to, and to an evident lavish expenditure in that department.

Some two years or more ago, a square furnace, a counterpart of that used by the Orford company, and a concentrating plant were erected at the Vershire mines, with a view to reduce expenses. Nevertheless, the property is now idle.

In the opinion of the writer, improvements, such as described, while well enough in their way, are not the proper course to render the mines profitable. They should be sought rather in so cheapening the transportation from the mines to the industrial centres, that the sulphur can be made available. In all of the mines connected with railways, the sulphur will pay for this transportation, and the roasted ore can then be economically treated in large central establishments located near the points of consumption of the copper produced.

About seven miles from the Vershire mines, the Corinth and Union mines are located.

They are part of the Vershire belt of pyrites, but the ore bodies are smaller, never exceeding a thickness of ten feet at their widest part, and a length of 150 feet on the strike. A section of the Union mine, shown by Fig. 3, is self-explanatory. Here, again, the quartzose hanging wall recurs, that we have found in Tennessee and North Carolina.

The strike of these beds is due north and south, pointing towards the Canadian mines at Capelton, which are, probably, on the same belt of rocks.

The ore mined at the Union mine was carefully broken and sorted, and then shipped. Formerly, Baltimore purchased the ore, but more recently, the property was sold to the Vershire company. In 1876, the Union mine shipped about 175 tons of ore per month, assaying 9.5 per cent. of copper. The cost of hauling to railroad was $3.00, and thence to Baltimore $9.50, an aggregate of $12.50 per ton. This left little or no margin. Together with the Vershire mines, the property is now idle.

Again taking up our northward march, we see how the Eozoic formation expands, in the New England States and Canada, and covers a wide range of country. The beds of pyrites are no longer confined to a narrow strip, striking northeast and southwest, but are scattered over a large territory. From Anthony's Nose,

almost due north, one range of deposits seems to extend, embracing Vershire, and the Capelton mines of Canada. Another series of beds takes a more easterly course, beginning with the Davis mine at Rowe, near Charlemont, Massachusetts, and including the Milan mines in New Hampshire, Blue Hill mines in Maine, some small beds in New Brunswick and Cape Breton, and the very considerable ones of Notre Dame Bay in Newfoundland.

There remain to be described only the Massachusetts, New Hampshire, Maine and Canadian mines.

Taking them up in the order of our march, we come first to the Charlemont mine in Massachusetts, or, as it is now known, the Davis mine.

The Davis Mine.

This body of pyrites belongs to the true iron pyrites, or iron bisulphide beds. It has a foot-wall of micaceous schist and hanging of chloritic slate. The ore-body has been developed for a length of 600 feet and depth of 300 feet, and exposes a vein, 9 to 21 feet thick. The mine is only 4 miles from the railroad at Charlemont, distant 60 miles from Troy, 119 miles from Boston, and 108 miles from New Haven. By reason of this proximity to markets, it is now profitably wrought, although containing almost no copper.

Average analysis of the ore is as follows:

Sulphur,	48.31 per cent.
Silica,	3.00 "
Copper,	1.41 "
Zinc,	none
Arsenic,	"

The capacity of the mine is some 3000 tons per month.

At one of the works using the ore, viz., the Merrimac Chemical Company, the kilns burn 800 pounds Davis lump ore in twenty-four hours, and reduce the sulphur to 3 per cent. Davis fine ore is consumed at the rate of 1300 pounds per furnace, but about 1 per cent. more sulphur, that is 4 per cent., remains in the residues.

The company, operating the mine, is doing a steady and increasing profitable business, even though the ore is valuable only for the sulphur it contains. Their success again emphasizes the necessity of utilizing the sulphur of the great Southern deposits to render them profitable.

The Milan Mines,

Situated in Coos County, New Hampshire, have been worked continuously since the middle of 1881.

The general character of the ore bodies is identical with the many described. The lenticular masses are 14 to 40 feet wide, and contain bands of blende and copper pyrites, and also occasionally some bunches of galena. Blende is commonly found near the walls, and diminishes somewhat in depth.

Average analyses of the pyrites by Amory Austin, of Boston, in 1883, were as follows:

Sulphur,	48.48 per cent.
"	49.30 "
"	50.29 "

The ore is claimed to be free from arsenic. This claim, indeed, is made by all the pyrites producers, yet acid made from pyrites invariably contains more or less of this impurity. The mine is only 102 miles from Portland, and connected with it by the Grand Trunk Railway.

The ore at the mines is sorted into three classes, which are as follows:

Sulphur ore,	75 per cent. of total mined.
Copper ore for smelting,	10 " " " "
Smalls,	10 " " " "
Waste,	5 " " " "

The No. 1 sulphur ore assays as follows:

Sulphur,	48.0
Iron,	43.0
Copper,	1.6
Zinc,	1.5
Silica,	5.0
Arsenic,	traces
Total,	99.1

Ore containing as high as 10 per cent. blende is sold as No. 2 sulphur ore. By reason of the formation of sulphate of zinc, such ore cannot be burnt clean.

The selected copper ore is smelted on the spot, and the resulting argentiferous matte is shipped.

The Blue Hill Mines of Maine

Created a considerable stir in the mining world of Boston some five years ago.

Copper ore only a few miles from deep water presented an inviting field. The Douglass and a number of other mining companies were organized, and work on a large scale begun at Blue Hill. The ore beds are of the "slate-ore" variety and quite lean. Both concentrating and smelting became necessary to beneficiate the ore, and a very extensive plant of this description is erected at the Douglass mine.

In depth all the ore beds have pinched, indeed they are faithful counterparts of the occurrence of these ores elsewhere.

The percentage of sulphur in the raw ores is too low to admit of working for acid, and at the present price of copper the future success of these beds, notwithstanding proximity to the ocean, is quite problematical.

The Coxheath Mines of Nova Scotia, recently discovered and claimed to be of some magnitude, are said to be similar to the deposits elsewhere.

Newfoundland Mines.

The Island of Newfoundland is the furthest northeast promontory of the Eozoic Rocks. Beds of pyrites were discovered in these rocks about the year 1864, almost on the ocean, in Notre Dame Bay. The beds occur in chloritic slate which, in places, is underlaid by a steatitic bed of rock, some 7 or 8 feet wide. The presence of the ore was indicated by ferruginous croppings, although the great masses of gossan found in the more southern ore beds are wanting.

The most prominent localities, all of which have been largely wrought, are Betts's Cove, Little Bay, and Tilt Cove. At Betts's Cove large operations began in 1874, when 8000 tons of ore were shipped. In 1876–77 and 1878, the shipments were 18,000, 44,000 and 25,000 tons, respectively. By reason of the proximity to the ocean, it became possible and profitable to sort the ores up to 7 to 9 per cent., and ship them to Swansea for treatment. There they brought the highest market price, as they were quite free from arsenic and antimony.

In 1878, cupolas were erected at Betts's Cove to matte the lean ores. The matte was sold as such. The lenticular masses of ore

at Betts's Cove reached a thickness of 60 feet. A width of 4 to 12 feet was, however, more common. The ore-bearing strata had a thickness of 250 feet, a northeast strike, and a dip of 45° to the north-northeast. White quartz, in places, cut off the ore, and indeed the description of the Ducktown deposits applies here, as elsewhere.

In some of the localities, sandstone is found near the ore beds. Undoubtedly, this sandstone is of a later geological age than the chloritic schists. At Ducktown, such sandstone is found about 250 feet from the Mary mine ore bed.

After the exhaustion of the principal ore masses at Betts's Cove and elsewhere, these mines languished until within the last few months, when new developments reported to have been made have led to a revival of mining. Over one hundred miners are now employed at these mines.

The Capelton Mines, Canada.

These mines are situated within a short distance of the railroad near the American line in the County of Sherbrooke, and have for several years been worked on an extensive scale. The openings are on a number of lenticular masses of pyrites, averaging 10 feet in thickness, reaching a maximum of 30 feet and a length of 300 feet on the strike. The mines are worked by two parties, both Americans; one the Orford Copper Company and the other the well-known firm of Nichols & Co. Their aggregate product is about 200 tons per day, and the whole product is now shipped by the Passumpsic Railway to New Haven and thence by vessel to the works of the two firms in New York harbor. The Capelton mines produce more copper than any of the many pyrites deposits described. They are being actively wrought and both their contents in sulphur and copper beneficiated.

The ores were first treated by the Orford Company on the spot. They were thoroughly roasted in heaps and treated in three large Raschette furnaces, built entirely of brick, and with a fore-hearth into which the molten material discharged continuously.

The raw ore at the Orford works analyzed on the average as follows:

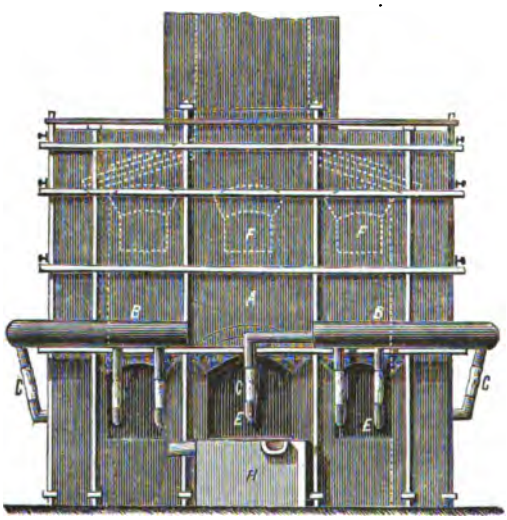
Copper,	5.0 per cent.
Iron,	35.0 "
Sulphur,	40.0 "
Insoluble residue,	20.0 "

The pile roasted ore had its sulphur reduced to 6 or 7 per cent. and was smelted in the furnaces already alluded to.

The product, a matte of 40 per cent., was shipped and sold. While the pile-roasting presented no novelty, a great step in advance was made in the matte-smelting by the use of a large Raschette furnace measuring 4 feet x 11 feet at the tuyeres. The height from the tuyeres to the charging floor was 8 feet, and the furnace was blown through fifteen 6 inch tuyeres.

Figures 26, 27, and 28 represent respectively a front elevation, section at tuyeres, and cross-section of the Orford furnace: D is the shaft of the furnace; A the walls of firebrick backed by red

FIG. 26.

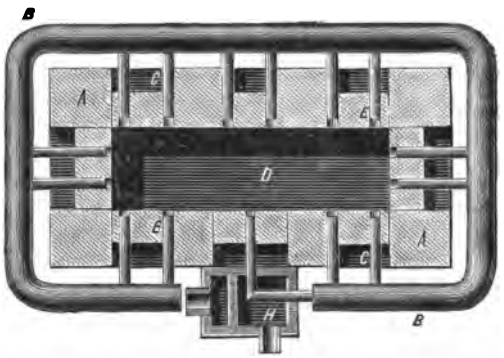


brick and strongly bound by railroad iron; F the three charging doors; E the tuyere arches with the wind pipes C, two in each arch, excepting only the breast or front arch.

The bottom of the furnace G is prepared by semi-fusion of quartz mixed with 5 to 10 per cent. of copper slag in place. This bottom, similar to that used in reverberatory furnaces, answered very well. All the molten material discharges continuously into the fore-hearth H, which is made of iron plates bolted to the buck staffs of the furnace. The fore-hearth is divided into two parts by an arch of fire-brick shown in the plan B. The matte passes under this arch and is discharged by the lip on the side of the fore-hearth.

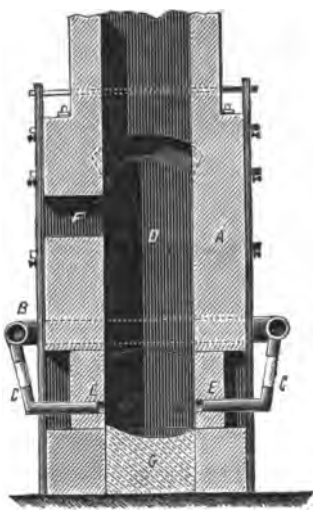
The slag accumulates in the large half of the fore-hearth and finally overflows by the front lip or slag-spout bolted to the fore-hearth. Both halves of the fore-hearth are kept covered with char-

FIG. 27.



coal or wood. With proper attention, and the necessary rapidity of smelting, the two spouts will discharge a continuous stream of slag and matte. The capacity of this furnace is 100 net tons per

FIG. 28.



day, and the fuel consumed (gas-house coke) amounts to 15 per cent. to 16 per cent.

It is the practice in this class of furnace to simply allow the

furnace to stand if the walls become too thin by the cutting action of the slag. Such slags made in a brick-lined furnace, must, of necessity be more silicious than if smelting is conducted in a water jacket, where erosion is impossible.

At the Orford works slags were indeed attempted to be made with as high as 45 per cent. Silica. Analysis of this slag proved the almost entire absence of copper, the cyanide test giving only a trace. But the slag was so tough that it became impossible to keep the furnace in good order, and the silica was reduced to below 40 per cent. With a slag carrying such a percentage of silica the copper rarely exceeded 0.5 per cent. and often fell to 0.2 per cent.

About three years ago the Canada works were abandoned by the Orford Company and works erected in New York harbor. Since that date the pyrites has been shipped to New York and first burnt for acid. Kilns with grates are used for the coarse which burn 1000 lbs. in 24 hours and reduce the sulphur to 3 to 4½ per cent. The fines are burnt to about 4 per cent. sulphur.

The roasted ores are now rarely used alone by the Orford Company but fluxed with other ores of various kinds. Smelting is however carried on in furnaces as described. As an experiment a water-jacketed Raschette furnace was tried, measuring 17 feet by 42 inches. While in operation it melted at the rate of 250 tons per day and that with the blast gates only partially opened. The furnace was found to have many drawbacks, principally in blowing over the fine ore into the flues, and has been abandoned. If the width of the furnace had been built in proportion to the length better results would undoubtedly have been attained.

The ores from the Albert mines at Capelton are shipped to the Laurel Hill Chemical Works in Long Island City for treatment. They are there roasted in kilns and smelted for matte. The furnace at the Laurel Hill works is unique in its construction, and originated with the manager, Mr. J. B. F. Herreshoff, to whom the writer is indebted for the following notes:

The Albert mine ore has the following composition, according to an analysis by L. Pitkin.

S,	38.15
Cu,	5.57
Fe,	32.64
Pb,	0.36
Zn,	3.09
Mn,	0.10
Al_2O_3,	2 93
CaO,	0.98
MgO,	0.30
SiO_2,	13.79
	97.91

The average of the ore is lower in copper by about 1 per cent. From the analysis the following approximate mineral composition is deduced:

Copper iron pyrites,	72.00 per cent.
Galena,	.41
Blende,	4.62
	77.03

The gangue is a mixture of silicates of lime, magnesia, alumina, iron and manganese, and some free silica (about 7 per cent.). There is an excess of iron in the analysis which is probably present as magnetite. The ore, as delivered on the dock at the Chemical Works by vessels from New Haven, is largely in lump. The coarse is burnt in kilns for the manufacture of acid. A ton of ore yields a trifle less than a ton of acid. The clinkers contain 2.5 per cent. of sulphur and are smelted in the Herreshoff water-jacket furnace for matte. Enough fine, raw ore is added to bring up the contents of the charge to about 7 per cent of sulphur. No flux is used, the charge being self-fluxing.

Gas-house coke is used as fuel. With these conditions a slag having the following composition results:

SiO_2,	per cent.	29.24
Fe,	"	41.66
Al_2O_3,	"	8.30
CaO,	"	trace
MgO,	"	none
Cu,	"	0.65
S,	"	0.82
Pb,	"	0.17
Zn,	"	2.47

The resulting matte averages as follows:

Cu, ...	per cent.	51.43
Fe, ...	"	22.75
S, ...	"	21.10

The above analyses were made by C. H. Joüet.

Matte running as high as 70 per cent. copper has been made without any trouble, and in the present practice the copper matte rarely drops below 50 per cent. metal. Comparing this with the Ducktown and Ore Knob practice of repeated roasting and smelting to obtain the same result, the evident great economy will strike the observer. The writer is not aware of any practice anywhere else

FIG. 29.

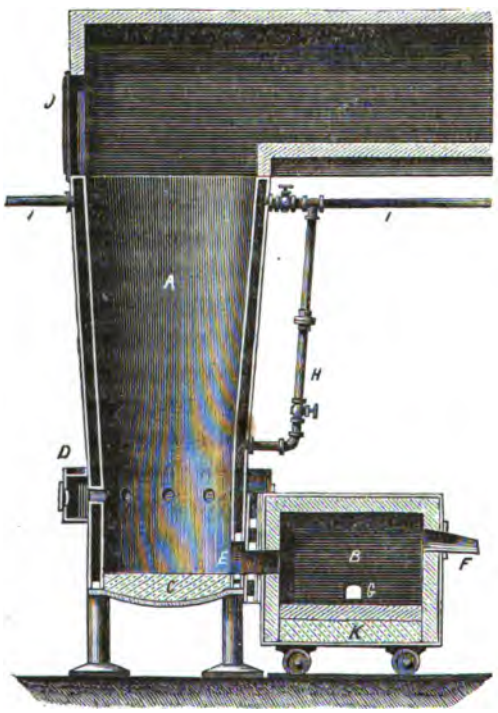
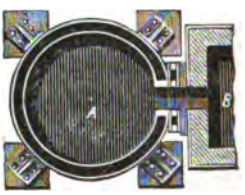


FIG. 30.



which, in a single operation, will produce so high grade a matte from a 5 per cent. ore, with equal economy both in cost of the operation and losses of copper in the slag.

In a former portion of this paper the Herreshoff furnace has been referred to as belonging to the type of German "spectacle" furnace, that is, all the molten matter is continually discharged

from the furnace. The improvements on the original have been such, however, that the principle only remains, and the furnace bears but little resemblance to its prototype. See Figs. 29 and 30.

In place of the old brick furnace-shaft, a water-jacket *A* is used. The forehearth *B* is placed on wheels and can be readily removed from the front of the furnace. The molten material flows direct from the shaft through the opening *E* into the forehearth. The joint between the two is readily made or severed, for it consists simply of two water-jacketed faces of iron which are placed in contact by moving the forehearth *B* against the jacket *A*. The cooled iron surfaces immediately chill any matte or slag liable to run between them, and make a perfect joint between furnace and forehearth. The first furnace erected was round, and 48 inches in diameter. Figures 29 and 30 illustrate its construction.

The water-jacket *A* rests on four posts attached to the bottom by brackets, as shown in the cut. *C* is fire-clay rammed into the bottom on a supporting cast-iron plate, fastened to the furnace; *D* is the wind-box; *H*, the entrance of cooling water; *I*, the exit of same; *O*, the charging door; *B* the rectangular forehearth with fire-brick sides, top, and bottom, held by an iron plate-casing; *K*, a layer of slag-wool; *F*, the cinder lip; and *G*, the tap-hole for the matte in the forehearth. The second furnace, built at the Laurel Hill works, was of the same circular shape, but 60 inches diameter. Blast was furnished at the usual low pressure by a Baker blower, and trouble was experienced by the failure of the blast to reach the center of the furnace.

The last furnace erected, and the one now in use, shown in Figures 31, 32 and 33, is rectangular in shape, with the corners rounded, and the lines between the corners slightly curved or of convex shape. The height is ten feet, width 3′ 7″ at the bottom and 4′ 7″ at the top, by 6′ 4″ length at the bottom, and 7′ 4″ at the top. The water-jacket is exceptionally narrow, having a water space of only 2 inches.

Referring to the cuts, *A* is the body of the furnace; *B*, a ring 2 x 2 inches, to which the plates of the water-jacket are riveted. At the top, *C*, the outer plate is flanged 2 inches, and the inner plate 4 inches, and the flanges then riveted. The bottom of the furnace *E* is a disked cast-iron plate 1 ½ inches thick, fastened to the ring *B* by tap bolts. This permits the dropping of the bottom if required. The legs *F* are bolted to the ring *B* on the outside of

FIG. 31.

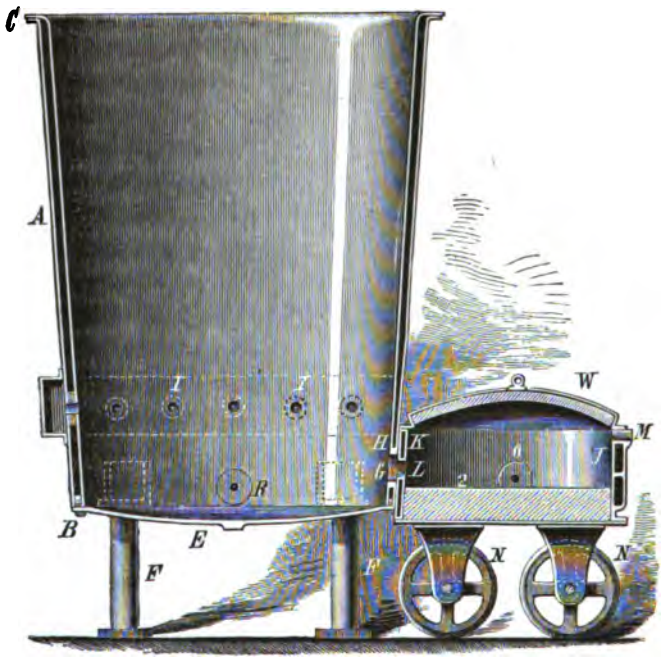


FIG. 32.

FIG. 33.

the furnace, thus not interfering with the dropping of the bottom. The hole *G* is the outlet of the furnace for both slag and matte. It is 9″ high and 7″ wide, and made by riveting the wrought-iron frame *H* into the shell of the furnace. The furnace is blown by 13 tuyeres, 5 on each side and three on the back. They are placed 26″ above bottom plate and are 2 inches in diameter.

The construction of the furnace proper is practically identical with that of the original round furnace, but the forehearth is considerably changed. In the round furnace (see Fig. 29) the forehearth was floored with a layer of slag-wool and brick as described. A brick lining was also used. The bottom of the brick lining was some 12 inches below the outlet from the jacket. Experience proved that this bottom invariably chilled to a level with the bottom of the opening to the furnace. The cutting of the brick lining at a higher level also gave occasional trouble. Both these faults are avoided in the present construction. The former by raising the forehearth on high wheels *N* and making the floor of the bottom lining within 2 inches of a level with the bottom of the inlet *L*. The latter, by entirely casting aside fire-brick lining and depending on the circular cast-iron water-jacket *K*. The tap-hole *R* in the shaft of the furnace is used only when blowing out to tap the furnace clean, or, sometimes, for such small quantities of black copper as may be accidentally made. In the forehearth the tap-hole *O* is the one commonly in use. It is made of copper, bolted to the iron body of the forehearth, and is water-jacketed similarly to the "Lürmann" slag tuyere of iron furnaces. The manner of operating it is also similar. *M* is the slag spout, *W* a brick-lined dish-shaped movable iron cover of the forehearth. When smelting, the well or forehearth is wheeled up against the furnace, as shown in the cut, and a very small amount of wet fire-clay is placed on the iron faces surrounding the holes *G* and *L*, in order to make a tight joint between them.

In practical operation after the furnace has been properly charged the blast is let on. The first cinder collects in the bottom of the furnace shaft proper, and accumulates until it reaches the holes *G* and *L*. It then overflows rapidly into the forehearth, carrying matte with it. In a short time the level of the molten material rises above the top of the hole *L*, and from that time onward the blast in the furnace can no longer blow out through *L*, and is completely trapped. Owing to the pressure of blast the level of molten

matte and slag in the forehearth is several inches above that in the furnace proper. Eventually the slag-lip *M* is reached by the cinder, which then overflows quietly. Matte is tapped periodically from the tapping notch *O* without stopping the furnace. Matte is never allowed to accumulate until it overflows at the slag-lip, the practice being to tap at stated intervals. The notch *O* is opened by a small steel bar, and pure matte, to the amount of about 1000 pounds, is allowed to run off. During this operation the level of the molten slag in the forehearth falls, but not sufficiently to admit of blast escaping through *L*. By the simple insertion of a small clay stopper the matte is stopped before cinder appears, thus avoiding all cinder picking. The whole process only occupies a few minutes, and is so perfect that for months a miss in tapping or closing up has not been made.

The large amount of molten slag and metal in the forehearth greatly facilitates a clean separation, as the slag analysis clearly shows. The high percentage of the matte, made without trouble from ironing, is entirely due to the great rapidity of the smelting. Standing at the charging-floor the charge sinks visibly while watching it, and is exposed so short a time to the action of reducing gases, that the iron is slagged before reduction, and thus ceases to be the obstacle to a rapid concentration of copper in a high-grade matte, which metallurgists usually consider it.

The following data of work, done by the different sizes of furnace, speak for themselves. The saving in fuel by the larger furnaces is apparent.

48 inch Furnace.—Average charge per diem.

Roasted ore,	51.7 tons.
Raw Fines,	4.3 "
Sand,	2.8 "
Total Stock,	58 8 "
Coke,	11.5 tons or 20 per cent.

60 inch Round Furnace.—Average charge per diem.

Roasted Ore,	76.0 tons.
Raw Fines,	8.0 "
Sand,	12.0 "
Iron Slag,	9.5 "
	105.5 "
Coke,	17.0 tons or 16 per cent.

Rectangular Furnace.—Average charge per diem.

Roasted Ore,	76.8 tons.
Raw Fines,	13.2 "
Sand,	5.3 "
	95.3 "
Coke,	17.4 tons or 18 per cent.

The cost of smelting in the large Herreshoff furnace is very low. The total employés per diem are ten men, and with gas-house coke at $2.50 per ton, and repairs exceptionally low, the total cost per ton of ore cannot aggregate 80 cents or on the ton of 50 per cent. matte about $10, or 1 cent per pound of ingot.

The practice here described, represents the most advanced state of the art in America. With an extensive general experience the writer believes it to be the best in use anywhere for the treatment of low-grade pyrites under similar conditions. While it may not save as large a percentage of copper (and silver) as the wet methods practiced in Europe, and to a limited extent in America, it is certainly better adapted to American conditions, for the cost of the plant is but a small fraction of that of the plant required for wet methods.

GRAPHIC TABLES FOR THE REDUCTION OF STADIA MEASUREMENTS.

BY BAILEY WILLIS.

THE first results obtained in stadia surveying are distances on the slope and vertical angles; it is usually desirable to reduce these to horizontal distances and differences of elevation.

Let d equal the horizontal distance for a known slope distance s, and a vertical angle θ, and h be the corresponding difference of elevation; and let c equal the distance from the centre of the instrument to the objective, plus the focal length of the telescope. Then, if the rod be held plumb, the relations are:

$$d = s \cos^2 \theta + c \cos\theta \qquad (1.)$$

$$h = d \tan \theta \qquad (2.)^*$$

* See "Topographical Surveying," by J. B. Johnson. J. Wiley & Sons, 1885.

Outline of a Graphic Device for the Reduction of Stadia Measurements

Follow that curve which is nearest the given value of s to its intersection with OD, and read the value of d, interpolating between curves;

Seek this value of d on OS, and follow the corresponding ordinate to its intersection with OD; the length of this ordinate, in terms of the vertical scale, gives the value of h.

With a diagram of this character, having a horizontal scale of 25 feet to 1 inch, and vertical scale of 10 feet to 1 inch, angles subdivided to five minutes and curves for values of s, differing by 25 feet, 1200 sights were reduced to horizontal distances and elevations in ten hours, equivalent to one sight each half minute, the labor having been divided between two persons, one of whom read the field notes and recorded results, while the other read the graphic table.

The accuracy of the results will depend on the execution of the individual table. With that used as above stated, which was checked by comparing its results with calculation in different parts of it, distances were obtained to 1 foot, and elevations to one-tenth of a foot.

In equation 1 the constant c depends upon the instrument used. Hence, it is necessary to construct a table for the instrument, or to construct one for general use on the basis of $d = s \cos^2 \theta$, and add this correction $c \cos \theta$ afterwards.

Another form of this table might be constructed by laying off the values of d calculated for any fixed value of s as abscissas instead of as polar-ordinates, thus dispensing with the scale O D; but in this case the curves approximate closely to the vertical lines of the reticle, and it is difficult to follow the subdivisions of θ without the aid of the pivoted scale OD.

Or d might be assumed to be constant and s variable for each curve. Then s must be read by the scale OD, and the nearest curve followed to the base OS; in this case the curves would be concave in the opposite direction.

Another table might be constructed to present the differences between s and d for varying values of θ.

From equation 1 we get:

$$S - d = s\,[1 - \cos\theta\,(\cos\theta + c)].$$

If in this we place $s - d = 1$ foot, calculate s for different values

of θ, and lay off these values as polar-ordinates, we shall get a curve which is the locus of $s - d = 1$.

From a table made up of such curves, drawn at intervals of 1 foot or less, the difference, which enters into the interpolation in any other form of table, might be read directly.

For the sake of illustration the line $s - d = 2$ is drawn on the accompanying diagram.

THE CLASSICS AND NATURAL SCIENCE AT A GERMAN GYMNASIUM.

BY C. A. STETEFELDT.

THE Gymnasium at Gotha had the reputation—and may have it now—of being one of the best schools in Germany. If instruction in the dead languages, almost to the exclusion of everything that is useful in life, is to be considered a measure of merit, the above-named school fully deserved this praise. At the time I entered the Gotha Gymnasium, in 1852, it counted among its professors two great philologists—Rost and Wüstemann. Rost, the director of the school, was a remarkable man. Besides being one of the first Greek scholars of his time, he was a man of broad culture, and even possessed such business qualifications that he occupied the position of president of one of the most prominent insurance companies in Germany. In contact with the students he was stern and severe, but just. He even went so far as to take the part of a student against a teacher, if he found that the former was right. We highly respected him, but I cannot say that he was loved. He inspired fear, but not hate. If he was in his wrath we trembled, and no one would dare to oppose him. Woe to him who was called upon to recite and found wanting! Such piercing sarcasm, such crushing invective would be rained upon his head that the poor fellow felt completely crushed. The Primus of the class once being called upon to recite, made rather a sorry mess in translating some lines of the Iliad. Rost asked him abruptly: "What do men carry upon their shoulders?" The student, being at a loss how to reply, kept silence. Rost answered: "A head, but you carry a wash-basin."

Rost's translations of the classics were marvels of originality, force, and beauty, and it is to be regretted that they have never appeared in print. In that respect the lessons were highly interesting But there was too little of this. In one hour only a few lines of Homer, Sophocles, or Virgil would be translated, and the time was principally taken up with the dry syntax of a dead language, that left our minds void, and made us coincide with the monk's opinion that Greek was an invention of the devil.

In Selecta, the highest class, we had to write once a week a translation of German into Greek, dictated to us in German. The subject was invariably taken from a Greek author. Why this was done I do not understand, but my supposition is that even the best Greek scholar does not feel perfectly sure of avoiding mistakes. There were very few in the class who could accomplish this task without the most severe labor; hence, we worked on the co-operative plan. We had two students in the class who were well read in Greek literature, and rarely failed to discover the author from whom the pensum was taken. Then the original was copied, taking care to avoid peculiarities of the author. Of course those who had the reputation of not being well up in Greek, inserted as many mistakes as they saw fit. Once it happened that a blockhead copied the author verbatim, and he was found out. Such a storm was rarely witnessed in the Selecta.

Wüstemann was the great Latin scholar, and Cicero was his ideal. His culture was all one-sided, and outside of classical literature and Roman antiquities, his mind was a blank. He could not even speak his mother tongue with correctness and elegance. He was more at home in Pompeii and Herculaneum than in his native city. In his intercourse with the students he was as amiable and mild as Rost was gruff and stern. But there was much Ciceronian conceit and weakness in his character.

He amused us and we loved him. I cannot say that he taught us much Latin. It seemed to him almost a sin to translate Cicero's orations or the odes of Horace into German jargon. In case a student was called upon to recite and was not well prepared, he only needed to give a sign to the class. Thereupon we would commence to ask questions about Cicero in particular and about the Romans and Rome generally, and as he never failed to answer the most silly questions the hour passed most pleasantly. We asked him once how many hairs Cicero had on his head. With-

out considering the absurdity of such a question, he answered that Cicero was bald—there were not even the three hairs of the man of blood and iron. Upon being asked what language he thought the angels in heaven were speaking, he replied, with an injured look, "Latin, of course."

Subjects of great interest to us were the household affairs of the Roman ladies, and here Wüstemann was at home in every detail. He once gave to a number of friends a dinner à la Lucullus.

If we were bent upon real mischief, we would ask questions relating to natural science. In that respect Plinius and Aristoteles formed the only stock of his information. His trusting disposition and shortsightedness often subjected him to impositions. I remember how a tall student announced that he had memorized one of Cicero's orations, and asked permission to recite it to the class. Boldly he mounted the rostrum, took an edition of Cicero out of his pocket, and read the oration with grand effect, to our amusement. After he had finished he received a high compliment from the Professor. We were all sad when we followed Wüstemann to the grave.

The other professors of Latin and Greek were all of that dry quality that is characteristic of the majority of philologists of a German Gymnasium, and classical schools generally. They never entered into the spirit of the author, but confined themselves to dry technicalities of the text. Many of us actually hated these lessons that seemed to be intended more to destroy than to awaken our intelligence.

Carlyle, in "Sartor Resartus," gives a sarcastic account of this system of instruction, as follows: "My teachers," says Professor Teufelsdröckh, "were hide-bound pedants, without knowledge of man's nature or boy's; or aught save their lexicons and quarterly account-books. Innumerable dead vocables (no dead language, for they themselves knew no language) they crammed into us, and called it fostering the growth of mind. How can an inanimate, mechanical Gerund-grinder, the like of whom will, in a subsequent century, be manufactured at Nürnberg out of wood and leather, foster the growth of anything; much more of Mind, which grows, not like a vegetable (by having its roots littered with etymological compost), but like a spirit, by mysterious contact of spirit; thought kindling itself at the fire of living thought? How shall *he* give kindling, in whose own inward man there is no live coal, but all is

burnt out to a dead grammatical cinder? The Hintershlag professor knew syntax enough; and out of the human soul thus much: that it had a faculty called Memory, and could be acted on through the muscular integument by appliance of birch-rods."

Our instructions in German and history, in the hands of Professor Regel, were excellent, and we all admired and loved this talented teacher.

In religion, we read the New Testament in Greek, but those not on the first benches preferred to read a novel or prepare themselves for the next lesson.

In French, we had a French professor, and a good deal of fun we had with him.

Mathematics we had very little, and only those who had a special aptitude for this science profited at all by these lessons. I have observed that, with rare exceptions, those students who excelled in Latin and Greek were the most backward in mathematics, and such instruction in natural science as we had. Since from these students the professors of classics are recruited, it is not to be wondered that so little progress is made in improving the system of teaching in the Gymnasium. The instructions in natural science in the classes Selecta and Prima were confined to two hours of physics per week. They were abominable. The professors would read us the paragraphs from a text-book on experimental physics, but an experiment was never performed. Once a semester we were taken to a room called the "physical cabinet," where we beheld, awe-stricken, a collection of antiquated philosophical apparatus. We inspected it with the same curiosity that country-folks view a chamber of horrors, filled with instruments of torture from the time of the inquisition. There we were allowed to turn the crank of an electric machine, or beat the disk of an electrophor with a fox-tail. We were as ignorant about physics coming out as we were going in. But some of us went away with the impression that there were many things between heaven and earth of which our learned professors had never dreamed.

It is but just to say that since the Gotha Gymnasium has been combined with the "Realschule," the instructions in mathematics and natural science are carried on upon an entirely different basis, and more in accordance with modern requirements. I must finally state that the task imposed upon a student in a German Gymnasium is so onerous, especially as far as the classics are concerned, that

were he to perform conscientiously all he is asked to do, he would break down, mentally and physically, in a year, even with an iron constitution. Hence, for self-preservation, he is forced to deceive his teachers. Surely that must be injurious to the formation of character, and warp the morals of a weak mind. Why an intelligent people like the Germans allow such a state of affairs to continue is a mystery.

Such was the condition of affairs when some restless spirits undertook to supply such mental food as the school considered superfluous to our training. The impulse to this step was given by some students of the Gymnasium at Eisenach, that beautiful city in Thuringia, at the foot of the Wartburg, where the minnesingers once reigned supreme, and where Luther translated the Bible. The Eisenach Gymnasium was still more given to an exclusively classical training than that of Gotha. Its bigoted director held the opinion, and openly expressed it, that anything outside of the classics was not worth knowing; that the study of natural science was pernicious to young minds, and would break up the discipline of the school. When he discovered that students of the Gymnasium had formed a society for the cultivation of natural science, he threatened them with expulsion and demanded its immediate dissolution. But the brave fellows were not frightened by such threats. Persecution in a just cause produces redoubled zeal, and stimulates propagation of ideas. The immediate outcome was the formation of a society similar to that of Eisenach by students of the Gotha Gymnasium. On the 12th of June, 1855, six of us founded the "Naturwissenschaftliche Verein der Gymnasiasten zu Gotha." On the 14th of June the first regular session took place, and the president-elect, Blau, delivered an address—strange to say—in Latin. Such was still the ban the classics held over us! This practice, however, was not continued, and subsequent presidents of the Verein, like myself, considered German more appropriate. The experience of our fellow-students at Eisenach made us careful to keep the existence of our society a profound secret. Like a persecuted religious sect, we assembled privately. To do so for any length of time was not practicable. Hence, we concluded to reveal the existence of the Verein to the professor of mathematics and physics, the only one from whom we might expect encouragement and support. Now our by-laws had to be placed before director Rost. After two days of suspense he appeared in

the classes Prima and Selecta, and not only gave his approval but praised us for what we had done. He warned us, however, not to neglect our regular studies, and expressed a hope that a society would be formed for the cultivation of the classics also. We had good reason to be proud of our director. The Verein was now actually under the patronage of the Gymnasium, and what such an eminent man as Rost had sanctioned, could not very well be undone by any of his successors. The Verein had members and associates. Each member would select a branch of natural science that suited his taste, upon which he would deliver a course of lectures. We were all self-taught professors instructing each other. I, for instance, delivered a full course of lectures on experimental chemistry, with experiments and not without them, like our professor of physics.

Hofrath Jacobs, the renowned artist, whose son was one of the most active members of the Verein, and whose grandson is a member now, placed a room in his residence at our disposition, where we held our meetings. Here we assembled once a week, holding a session of two or three hours. Special meetings took place besides. We soon accumulated a library of scientific works, and collections of philosophical apparatus, minerals, chemicals, plants, etc., were started, many donations being received from patrons of the Verein.

At the end of the year 1855 our active members had increased to nine. A lively correspondence with our Eisenach fellow-students was kept up. Besides the regular courses of lectures on chemistry, physics, geology, mineralogy, and zoology, papers on special subjects were presented and new books reviewed. The anniversary of the Verein was always celebrated with great solemnity, and generally took place in one of the beautiful nooks of the Thuringia forest, where the members of the Eisenach Verein often joined us. Those members who had left the Gymnasium and entered the university would, during the vacations, attend our meetings and continue to further the interests of the Verein. This custom has been kept up to this day. On the 14th of September, 1869, the Verein celebrated the centennial anniversary of Humboldt's birthday with an oration. Upon Dr. Petermann's request a collection was taken up for the German North Pole Expedition.

In 1863 I emigrated to the United States, and in practicing my profession as a mining engineer in the West, I had completely lost

track of the Verein. Most agreeably was I surprised when, in 1880, I received in San Francisco a letter from Gotha containing an invitation to the celebration of the twenty-fifth anniversary of the Verein. The institution I had assisted in founding had now reached the respectable age of a quarter of a century.

The Verein exists to-day and is in a flourishing condition. It has a library of over four hundred volumes, and keeps the most important scientific periodicals. Its collections have grown wonderfully. Its members have access to the astronomical observatory at Gotha, where formerly the renowned Hansen presided. Of its twelve active members two are sons of old members. Professors of the Gymnasium, some of whom are old members, visit the weekly meetings occasionally, and are always present at extraordinary meetings and celebrations. Professor Otto Burbach, Curator of the Gotha Museum, is the special patron of the Verein, and lends a helping hand whenever necessary. The theory of the Eisenach Gymnasium director that the study of natural science is pernicious to young minds has found no favor in Gotha.

I cannot neglect to mention an incident at the Gymnasium that influenced the career of several members of the Verein. After Wüstemann's death Professor Berger became the principal instructor of Latin in the class Selecta. We had once a week disputations in Latin upon some thesis chosen by one of the students. When my term came I selected as my thesis: "The study of mathematics and natural science is superior in training the mind to the study of the dead languages."

Thirty years ago it was risky in Germany to hold such an opinion, and sacrilege for a student of a Gymnasium to express it. The class was struck with awe by such temerity, and only the members of the Verein applauded, while the professor, pale with rage, left his seat to begin and carry on the disputation in person. The Latin of the members of the Verein was like that of the monks of the Middle Ages, very forcible, but somewhat ungrammatical and lacking Ciceronian elegance. So far as the linguistic part of the controversy was concerned we acknowledged our defeat, but as to the logic we claimed the victory. The senseless rage of our professor was a strong argument in our favor. After this my position in the class became decidedly uncomfortable. The professors of the classics considered me a "black sheep," and feared that my revolutionary ideas might spread. I saw there was no

use in remaining longer at the Gymnasium, or to make an effort to pass the abiturium. At the end of the semester I left for the university of Goettingen, being well prepared, thanks to the Verein, to enter upon a study of natural science. Rost, who was a personal friend of my father, conceded that I was right, and dismissed me with a flattering testimonial. Professor Berger was right also. The seed I had sown bore fruit, and the precedent I had created was followed by three others, who left the Gymnasium also without taking its full course. One of them, I am sorry to say, became an apostate, and after tasting a semester of academical freedom, returned to the Gymnasium, repentant, to cram himself with more Latin and Greek, pass the abiturium, and return to the university to study law. I am sure, however, that the knowledge he acquired through the Verein has benefited him in his practice as a lawyer.

My readers may now very properly ask the question: What good has the Verein done? In what way have its members profited by it? Have they made successful careers in life? Would it have been as well if the Verein had never existed? To understand an answer to these questions fully it is necessary to be familiar with certain phases of German life. The students who are educated at a German Gymnasium, and take the full course of its instructions, generally go to the university to study either theology, law, philosophy and languages, or medicine, with the expectation of entering the government service. In a small principality like Gotha, the number of men who aspire to fill government positions is in excess of the demand, much more so than in the German empire as a whole. Ministers, teachers of high-schools, officers of the civil service, lawyers and physicians, prefer to send their sons to the universities to choose one of the professions, or "Brodstudien" in place of educating them for a commercial or technical career, partly because it is the custom, partly through mistaken pride. Any movement that would direct even a small number to leave the beaten track, and induce them to become chemists, geologists, metallurgists, mining engineers, etc., would not only improve the individual chances of these men in the race of life, but leave more breathing-room for the others, especially as the former were forced to seek employment abroad, the limited industries at home (I mean in the principality of Gotha) being insufficient to utilize their services. This is exactly what the Verein

has done. We have not been so fortunate as to produce a Humboldt or a Darwin, but the Verein counts many old members who occupy honorable positions and who rank above the average. In proof of this it was my intention to give a complete list of the old members of the Verein and state what has become of them. This, however, I find to be a most difficult task, as most of them have left home and are scattered all over the world. Even at the twenty-fifth anniversary of the Verein only a portion of the old members could be reached. This in itself is a proof that the Verein has been instrumental in inducing its members to break the narrow barriers of the fatherland and seek a field of usefulness in the world at large. That the existence of the Verein has been the means of influencing the career of many of its members, and has widened the intellectual horizon of all of them cannot be denied. That it still continues to exert this influence is evident from the fact that nearly all the members of the Verein who left the Gymnasium the present and the past year have chosen a study, and will follow professions connected with natural science. Professor Burbach informs me that by 15 of these students the following selections have been made: 3 natural science, 4 medicine, 1 forestry, 3 civil and mechanical engineering, 2 engineer corps of the army, 1 modern languages, 1 theology and philosophy.

ON FALSE PSEUDOMORPHS OF QUARTZ.

BY WM. EARL HIDDEN.

In the older works on mineralogy mention is made of "pseudomorphs of quartz after barite;" "pseudomorphs of quartz after calcite; and of "pseudomorphous-quartz," as occurring in Rutherford County, N. C. In all the mineral collections where the writer has seen specimens from this locality, they have been found labelled as above noted.

Opinions seem to have been divided between the identity of these forms with barite and calcite, though I have often heard them stated as representing the form of "an unknown species."

The specimens are conspicuous for their ever varying and un-

symmetrical forms, their inconstant angles, and for being almost invariably hollow, like geodes. The best specimens from Rutherford County, N. C., were found by Col. Twitty previous to 1870, and as early as 1850 the locality was well known. It had a local fame based upon the not unfrequent discovery of "water-bearing crystals" among the irregular quartz masses, which had a ready sale and were the cause of whatever development the locality ever received.

The writer visited the place in 1879, and had a little work done there in uncovering the vein. Nearly every vestige of former work having quite disappeared, a spot where a small stream crossed the vein was chosen, but my success was very meagre.

The surrounding country is largely gold-bearing, and is made up of surface disintegrated gneissic slates covered with vicinal drift, with frequent quartz veins appearing in masses or as outcroppings *in situ*. The vein of these pseudomorphs was gashed and cut in every possible direction, the gashes looking in places like knife thrusts or like marks left by a saw. Large masses, seemingly compact, would fall to pieces when lifted, separating uniformly along the lines of the gash marks into many pieces, all partly hollow and of unlike shapes. The vein seemed to be composed wholly of this kind of cellular, easily separable, quartz material. At the time of my visit I had become convinced that these irregular, apparently crystallized, masses were not a *true* pseudomorph, that they did not present the form of any mineral species, much less that of "barite" or "calcite." Opportunity to further study these forms of quartz did not present itself until lately, and meanwhile two new localities have been discovered. I should have mentioned that the first locality is a short distance northward from Rutherfordton, the county seat of Rutherford County. As late as 1881 it is credited with "pseudomorphous quartz after calcite." (See Geol. N. C., Chap. I., Min., Genth and Kerr, page 116. Raleigh, 1881.)

The second locality is situated in Iredell County, near Statesville, on the farm of a Mr. Crawford, the distance being about fifty miles from Rutherfordton in a northeast direction. It is noted for the abundance of the material there occurring, the large size of separable masses, the almost solid character of the crystalline (?) forms, and the non-occurrence (as yet) of fluid bearing masses. Otherwise the locality is similar to that near Rutherfordton.

The third locality and the best for specimens containing included water and air exists "in Upson County, Georgia, on the land of Joseph A. Cotten, Esq., three miles from Thomaston." Mr. Cotten (*l. c.*, November 24th, 1884), states that the original discovery was made in a deep hillside gully where the rains had worn a "washout" of 65 feet in depth. In this gully, as early as 1879, he had found large masses of cellular quartz that were fluid-bearing, but no small detached crystals (?) were found until about 1880. In all, Mr. C. thinks about a ton of large fluid-bearing masses have been exposed in this gully during the past five years. He remarks about the irregularity in form of the cells containing water, and their being lined with brilliant quartz crystals.

Now the similarity of these three occurrences is noteworthy, and points to a common origin for all such pseudo-crystal forms.

From a careful study of the rock *in situ* in North Carolina and many specimens from all the localities, I am forced to conclude that these forms of quartz are *pseudomorphs of the interstices between crystals* of some unrecognized hexagonal mineral that crystallized in thin flat tabular forms.

Sections of many of these water-bearing crystals (?) present an interior of bright transparent crystals, or of mammillary chalcedony; while the structure of the walls is semi-radiated from the exterior. Careful examination of the surfaces, particularly of the forms from Georgia, shows a bright series of triangular markings (angles 60°) on all sides. Now these markings are exactly what we would expect by the slow deposition of quartz on the basal pinacoid of a uniaxial crystal (hexagonal or rhombohedral), or of the deposition of quartz from solution in a vein filled up with meshed and netted crystals which being thin, presented only basal planes for contact surfaces. What the original mineral was is not shown by the specimens. The casts of crystal cavities in the larger masses show an unmistakable hexagonal prism with a large development of the basal pinacoid (these two planes identified by striations on the quartz), and this characteristic is persistent at all three localities.

A density determination of one small five-sided irregular crystal (?) that had walls barely one-tenth of an inch thick, and which was three-quarters filled with water, yielded 1.81, while another with thicker walls gave 2.16 (quartz being 2.67).

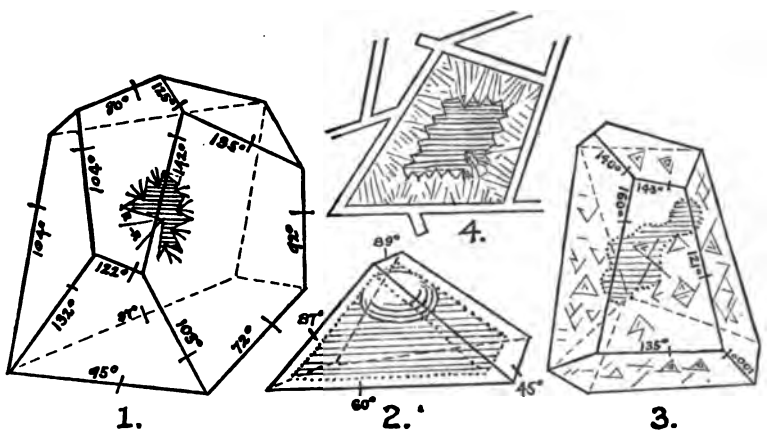
A careful test by Mr. J. B. Mackintosh of the fluid contents of

a small four-sided crystal (?) proved it to be only neutral water, and the bubble to be only air (not carbonic acid gas as was expected).

At the Georgia locality the chalcedonic specimens are frequently found and are more *durable* than those composed wholly of clear crystallized quartz. It is a matter of interest to note that specimens when first found, and noticed to contain water and a small bubble of air, will be seen in a few days to be gradually losing the water in some mysterious manner—the bubble all the while increasing in size. Not a few of these interesting pseudomorphs have within a few weeks entirely parted with their fluid contents. (As a remedy for this loss I recommend varnishing the specimen with a solution of collodion to which 5 per cent. of balsam of fir has been added.)

That the reader may arrive at a better understanding of these interesting forms of quartz, I have prepared an outline plate of crystals (?) from the several localities.

Fig. 1.—A nearly solid pseudomorph from the Iredell County locality. It weighs over a pound. It is milky-white quartz stained with iron oxide. It has the angles as shown in the drawing.



False pseudomorphs of quartz.

Fig. 2.—The specimen (natural size) from the Upson County, Georgia, locality having the sp. gr. of 1.81. It is composed of chalcedony. Its surface has much oxide of iron adhering to it and is rough, as if etched with hydrofluoric acid.

Fig. 3.—Same locality as last. Shows well the triangular mark-

ings of the mineral on which it was deposited. It is chalcedony, with an interior of crystallized quartz. Natural size.

Fig. 4.—From Rutherford County, N. C. Shows the gashed appearance of the quartz masses and the semi-radiated deposition of the quartz crystals in the original cavities.

(The ruled lines represent included water.)

In conclusion I wish to express here my thanks to Col. Twitty, of Rutherford; to Mr. Kestler, of Iredell County, N. C.; and to Mr. Joseph A. Cotten, of Thomaston, Georgia, for their generous gift of specimens, and also to Mr. J. B. Mackintosh, for his kindness in making the test of the fluid, as already noted.

I would only add that the Georgia locality is the most available at present, and promises the greatest yield of the much desired specimens having the fluid and air inclusions.

It is to be hoped that at no distant day deep work at one of these localities will discover specimens of the thin hexagonal plates, which left *the interstices between them only* and in the form of a mould of quartz.

THE REES RIFLE FOR MILITARY AND SPORTING PURPOSES.

BY FRANK REES.

THIS rifle is a departure from the old methods usually followed in the construction of the breech mechanism of rapid firing breech-loading firearms, and is designed for the purpose of giving to the soldier and sportsman an arm which shall enable him to fire at will consecutive shots in the shortest possible interval of time, and with the least possible effort or motion, and without the necessity of taking down from the shoulder or disarranging the aim while reloading.

A magazine is made use of, the same being placed beneath the barrel, and a revolving carrier is employed for receiving the cartridges from the magazine and elevating the same to the level of the barrel; a sliding breech-pin forces the cartridge out of the carrier into the barrel, and supports the same during the explosion and withdraws the cartridge shell. The revolving carrier simul-

taneously conveys down the spent shell and elevates another cartridge into line with the barrel.

One especial feature of the invention consists of a mainspring and chain belt and chain wheels and gearing, by which the carrier is revolved and the proper motions given to the other parts of the gun. This mainspring is wound with a key once for every fifteen to thirty shots, depending on the length of the spring; and the construction of the parts is such that when the trigger is pulled the arm is fired. Upon the second pull of the trigger the spring revolves the parts so that the spent cartridge is withdrawn from the barrel and lowered from the line of the barrel to the line of the magazine and ejected; a fresh cartridge is elevated by the carrier and forced into the barrel by the steadying breech-pin and held there firmly, and the parts are then in position for firing, hence the only movements necessary to operate this arm, by the person using it, is to pull the trigger to fire and to pull the trigger a second time to reload, alternate pulls of the trigger firing and loading, the power for all other purposes being derived from the mainspring.

The mechanism may be understood from the following description with accompanying plates.

The power necessary to work the respective parts is supplied by a mainspring H (Fig. 2) placed within and firmly attached to the stock, and which is wound up by a permanently affixed or detachable key or crank at any time previous to the loading of the gun. This spring drives a chain belt H^2 running over and revolving the chain wheel I (Figs. 1, 9, 13) which is provided with teeth or cogs to prevent the chain from slipping, and this chain wheel I is affixed to the shaft i and supported by the lock frame and revolves the tumbler K at the other end of the shaft i in the opposite direction to the movement of the hands of a watch. This tumbler K in the first half of its revolution draws the connecting link l and the breech-pin L (Fig. 1) backward out of the carrier G, and in the other half of its revolution forces the breech-pin forward again through the carrier, and in this way giving it a backward and forward motion within the supporting lock frame I^2. The rear portion of the breech pin is formed of a toggle link folding downward L^3 and L^4 and the front portion is cylindrical and hollow. The revolving carrier G contains two cartridge chambers (Figs. 8 and 9), one on a line with the magazine and the other on a line

with the barrel, these chambers being slotted at their sides for the passage of the link-pin l^1 of the breech-pin. When the tumbler K revolves it draws back the link l and the breech-pin L, bringing with it the empty cartridge shell from the barrel into the carrier by means of the extractor spring hook o^1 and supporting finger o^2 (Fig. 7), and the carrier then receiving a half revolution cuts the empty shell out from between the extractor and finger and lowers it to the line of the magazine, where it is pressed back out of the lower chamber of the carrier into the ejecting chamber M (Figs. 9, 12, and 14) by the next loaded cartridge coming into the carrier from the magazine, and from which chamber the empty shell is ejected as afterwards described. The carrier is given its half revolution by the three cog-teeth 7 (Figs. 15 and 1) on the inner surface of the rear part of the tumbler K coming into contact with the six toothed cog pinion g^1 (Figs. 9 and 10) which pinion is attached to the carrier shaft g, the front end of which shaft is firmly fixed into the carrier G. At the same time that the carrier makes its half revolution, lowering the empty shell in one chamber, the other chamber elevates the loaded cartridge received from the magazine to a line with the barrel, which is then forced into the barrel by the forward movement of the breech-pin L and supported by it during the explosion, the toggle links L^3 and L^4 which had been folded downwards being drawn up flat or straight as shown (Fig. 1). The momentum of the toggle links as they straighten out, carries the middle joint l^5 above the straight line between l^6 and L^3 and at that instant a cam i^4 on the revolving cam wheel I (Figs. 9, 11, and 13) comes under and supports and helps to raise the stud 25 (Fig. 11), which projects from the side of the toggle, above the line of the other two joints, so that when the explosion takes place the toggle joint l^5 is forced against the frame E of the gun and supported by it, and no part of the strain from the explosion is received by the other portions of the mechanism. When the tumbler K commences to revolve the cam i^4 moves forward from under and away from the stud 25 on the toggles, so that they are bent or folded downward by the action of the toggle spring l^7 (Fig. 1) between the toggles and housing E, and allowing the breech-pin to be drawn backward.

At the same time that the breech-pin is forced forward, it compresses the expansion firing-pin spring n^1 (Fig. 5) contained within the hollow portion of the front part of the breech-pin, by the action

of the sear hook n catching hold of the hook n^1 of the rear portion of the firing-pin n n and retarding it, so that when the breech-pin has forced the loaded cartridge home into the barrel the firing-pin spring is compressed and the firing-pin cocked ready for firing.

To fire the arm the trigger F^1 (Fig. 1) is pulled, forcing forward the trigger sear s, which in its turn forces forward the sear n^3, and lifting up the hook n^2 releases the firing-pin, which is projected forward and strikes the fulminating cap of the cartridge and explodes it. When the part 26 of the trigger sear has passed beyond the notch at 26 on the tumbler K and the firing has taken place, the sear s is held by this notch, so that when the trigger f^3 moves backward by the pressure of its spring f^1 the trigger latch s^1, by the action of the latch spring s^3, is dropped into the rule joint of the trigger which is hinged at 6, and when the trigger is pulled the second time the upper end of the sear s is forced further forward away from and clear of the tumbler K which is then free to revolve, drawing out the empty cartridge as described and tripping up the latch s^1 by striking its tail, and the part at 26 of the tumbler is arrested by the hooked portion of the trigger f^3, for the purpose of relieving the cog teeth 7 on the tumbler and the pinion g^1 on the carrier shaft from receiving too severe a shock. When the latch s' is tripped up it allows the sear s to be pressed back into its normal position, as shown in Fig. 1, by the action of its spring s^4, where the sear s is in position to arrest and support the tumbler K when it has completed its full revolution and the parts are all in proper position for firing.

After the empty cartridge has been passed back from the carrier into the ejecting chamber M it is kicked out during the next revolution of the cam wheel I (Figs. 13 and 9), the cam i^3 striking the projecting lever r, partially turning the shaft r^1 on which is attached the kicker 14, which comes behind the ejector P and forces it to eject the empty cartridge with great facility. The ejector P is hung on the shaft 12 of the door m and this ejector has a shoulder as shown (Fig. 12), which takes hold of the door and lifts it open when the ejector is thrown forward, allowing the empty cartridge to be thrown out. The door m then closes instantly by the action of the spring 31 pressing against the small crank arm 30 of the door shaft 12, to which shaft the door m is affixed. By pressing the door m further back than it is opened by the ejector P it remains open, and the cartridges can

then conveniently be passed in through the ejecting chamber M and through the carrier G into the magazine, when they force the follower up the magazine and compress the magazine spring until it is full. The pressure of the magazine spring delivers the cartridges into the carrier as needed, and forces the empty cartridges out of the carrier into the ejecting chamber as described. When the arm is being loaded and the cartridges are passed into the magazine, it is found to be advisable to have the last cartridge remain within the carrier, and not in the ejecting chamber, for which purpose a gate q (Figs. 14 and 13) is provided at the end of the kicker shaft q^1, which gate can be pressed inward by the thumb piece q^3 outside the lock frame, and this gate partially closes the front of the chamber M and prevents the loaded cartridge from passing back into said chamber. After the arm is fired the first revolution of the cam wheel I strikes the arm r, partially turns the shaft r^1 and throws the gate which is at the end of the shaft r^1 q^1 back into its normal position (shown in Fig. 14), so that the next empty cartridge shell is free to pass into the ejecting chamber. To prevent accidental discharges when the gun is loaded and cocked, a locking arrangement is provided. The finger piece t^3 (Fig. 1) is directly in front of the trigger and covered by the broadening of the trigger guard and within easy reach of the trigger finger. By pressing the finger piece t^3 in one direction the blocking piece t is turned in the path of the sear arm n^6 in such a manner as to block any movement of the trigger and prevent the arm being fired. By turning this finger piece t^3 in the other direction the cam arm t^4 on the upper part of the shaft t^3 unlatches the sear latch n^8, and at the same time coming in the path of the sear arm n^3 blocks any movement of it above the joint 27, so that the sear hook n^3 cannot liberate the firing-pin, but the end n^6 is free to move about the joint at 27, so that when the trigger is pulled twice the sear arm s is moved clear from the tumbler and the parts revolved for the purpose of withdrawing a loaded cartridge without firing it. The blocking piece t, as well as the gate q previously described, are provided with strong friction springs, so that they remain in any position in which these parts are placed.

Upon the carrier shaft g and behind the pinion g^1 is the square block i^3 (Figs. 9 and 10), and upon the inner surface of the tumbler K are the flat ribs 6 (Fig. 21), the purpose of these ribs and the square block being to stop the revolution of the carrier after it has

completed its half-revolution, and hold it firmly in its proper position, while the breech-pin is withdrawn from the carrier. This is accomplished very perfectly by one of the flat ribs 6 coming into flat contact with the flat sides of the square block i^3 after the last tooth of 7 on the tumbler has separated from the pinion g^1. Within the magazine tube is the follower, which, when the cartridges have all been withdrawn from the magazine, projects into the revolving carrier and locks it, preventing the carrier from revolving, and in this way immediately notifying the person using it that the arm must again be loaded.

When the breech-pin has been drawn back after firing, the extreme front end of the firing-pin projects from the end of the breech-pin, hence there would be a risk of exploding the loaded cartridge when the breech-pin was driven forward to force the loaded cartridge out of the carrier into the barrel. To prevent this contingency the set-back O (Figs. 5 and 7), is provided, which is pivoted in a mortise in the breech-pin L, so that when the breech-pin is drawn back the set-back comes in contact with the projection o^4, and is forced inwardly, and its inclined rear forces the firing-pin backward, and draws its point within the breech-pin. To prevent also any chance of firing the arm when the parts are not in proper position, a blocking-piece, f^6 (Fig. 1), is affixed to and projects from the trigger-hook f^3 so that when the tumbler K has completed its full revolution as shown in Fig. 1, the end of this arm passes under the last cog-tooth 7 on the tumbler, but if the tumbler K has not completed its full revolution, the cogs 7 are in the path of the blocking piece f^6 and the trigger cannot be pulled, and the gun cannot be fired while the parts are out of their correct position.

It should be said that the plates shown are somewhat defective, as they are from the drawings of the first model, which has been improved and parts and springs strengthened, and others done away with, the frame or housing E being a skeleton frame for the purpose of leaving all the parts exposed to view while experimenting. Two forms of chain-belts are shown, but the one in Figs. 1 and 3 is preferable, being stronger, and the cog-teeth on the wheel I and mainspring barrel projecting up between the double links hold the chain from slipping.

The advantages claimed for this invention are:

The ability to fire with greater rapidity than any gun now in

use, and with perhaps as great rapidity as it is possible to attain with an arm which can be carried in the hand and consistent with safety. One complete explosion must have time to leave the barrel before another cartridge is inserted.

The reloading of the arm without taking down from the shoulder and without other motion than a simple pull of the trigger, and without destroying the aim or taking the eye from the line of sight.

The automatic action of the arm, after the mainspring has been wound up and the magazine charged; requiring only to aim the piece and pull the trigger to fire, and to pull the trigger again to reload, and so on until the magazine is exhausted, alternate pulls of the trigger firing and loading, and allowing the person using it to concentrate all his effort on aiming.

The mechanism all being contained within the metal frame or housing, with no openings other than the trap-door and muzzle, the trap-door is always closed except in the act of filling the magazine or momentarily opened when firing. By allowing an empty cartridge to remain in the breech, or putting an empty cartridge tompion in the muzzle, the arm when not in use can be corked up, and made practically sand and dust and almost waterproof.

The arm being hammerless, there is no danger from accidental discharges, which quite frequently occur from the hammer catching with guns that employ projecting hammers.

A safety-lock directly in front of the trigger, covered by a broad guard, and which can be moved quickly and conveniently for locking by the trigger-finger, and unlocked while raising the gun to the shoulder without losing time, and when locked allowing the gun to be carried loaded and cocked with safety.

A magazine "cut-off" or gate, by means of which, by a simple pressure of the thumb, the magazine, after being filled with cartridges, can be disconnected or cut off from the loading mechanism, and the magazine held in reserve, allowing the arm to be used as a single-firing piece, and in any case of great emergency the magazine can be instantly connected again by pressing back the thumb-piece and connecting the magazine again with the loading mechanism, so that the arm can then be fired with great rapidity.

The action of the arm is easier and simpler for single firing than any of the single-firing breech-loading rifles, and at the same time

it is capable of being used as a rapid firing or hand Gatling machine gun.

The use of the elbow-joint principle to withdraw the exploded cartridge-case, increases the power of the mainspring to such an extent that it is sure to withdraw the empty shell even when fouled, so that it is easily ejected.

The working parts of the breech-mechanism are all attached to the lock frame, so that by removing the side plates and a few screws, all the breech-mechanism can be removed in one compact body, and cleaned and replaced, without altering their proper adjustment or reassembling.

The object of this method of constructing the arm, is to have as many of the motions necessary to reload all going on at the same instant, taking a minimum of time to accomplish the whole operation. It will be seen that in one complete revolution of the tumbler everything necessary is accomplished, and as the action of the mainspring is practically instantaneous, we might expect as a result, *instantaneous reloading*.

This is exactly the result attained, and the reloading is so rapid and perfect that it is almost impossible to see any motion at all. This can only be properly appreciated after witnessing the actual firing of the arm.

With such an arm it would seem almost impossible for armies to come into close or hand to hand conflict. A body of troops equipped with it and entrenched, would be extremely difficult to dislodge by assault, for while the enemy were at a distance it could be used with great rapidity and effect as a single-firing gun, the magazine being held in reserve until the assailants had gotten within 50 feet, or so close that accurate aim would be unnecessary, and the magazine then being called into use, a raking fire of 16 shots could be fired by each soldier before the 50 feet could be traversed.

In our late civil war, probably the average soldier who served four years might have been engaged in actual battle seven days, and assuming that out of those seven days he was in a position where he could actually aim and fire at the enemy within range not more than 12 hours, it would therefore seem that an arm which, after being carried around for *four years* at great trouble, could do the *most effective work in those 12 hours*, would be of the greatest importance. It has often been said that the real effectiveness of

the bayonet lies chiefly in its *moral power*, as they have seldom been crossed by large bodies of troops, "perhaps 7 or 8 times in the history of modern wars." Few soldiers can be made to stand firm and resist the bayonet charge. If, then, the moral power of a weapon is of such value, what would be the moral power of a body of troops entrenched and armed with a rifle that could be used effectually as a single-firing arm at long range and the magazine reserved until the enemy were close enough to see the white of their eyes, when 16 volleys could be fired in 8 seconds. It would seem impossible to get men to face such a volley.

For sporting purposes it gives the sportsman every chance to secure his game, and enables him to fire a second or third or more shots when the first has missed, without material change of aim and without the necessity of taking down from the shoulder to reload. With an animal on the run, a half-dozen shots might be gotten where only one could be had with the ordinary single-firing rifles—the especial point of advantage for a sporting rifle being in the fact that the second shot can be instantly fired when the first has missed without destroying the aim by reloading. In all guns which require being taken from the shoulder to reload, the aim is destroyed and it is the securing of the second aim that requires as much if not more time than the actual reloading. In shooting at herds of animals it gives a chance to secure a number before they can get beyond range.

The breech-mechanism is applicable to any form of barrel, size of cartridge or bore, and can be adapted to the standards of the different governments. The breech-pin is locked in such a manner that the entire strain of the explosion is received by the breech-pin and carried by it to a small portion of the frame, so that all other parts need only be made strong enough to properly perform their respective motions. By making use of a carrier revolving upon a fixed axis, there are no openings necessary to the gun, and the motion is exact and true and not liable to clog, and we consider it a great improvement on the square brass carrier, working in a large slot open above and below, which is used in some forms of magazine rifles.

The safety-lock would be an improvement on most other guns, and would render accidents much less frequent.

The experimental gun which has been used for designing this invention has demonstrated the fact that it is not liable to get out

Fig 3 [redacted]

Fig 2

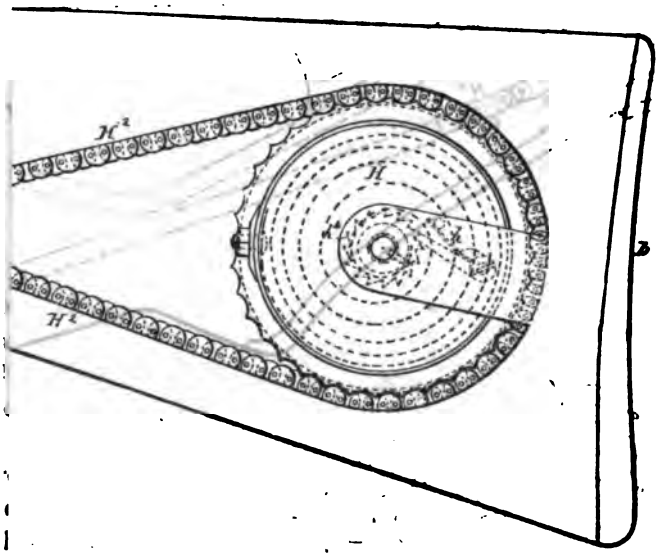


Fig 13

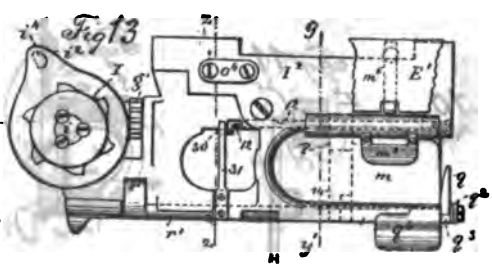
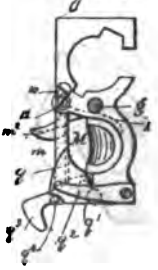


Fig 14



of order when properly made, and will stand an immense amount of use, the model having been fired thousands of times with ball cartridges, and the parts showing very little wear, working smoother and easier than when first constructed. The capacity of the arm is two shots per second, and at a recent trial before the officers of the 7th Regiment, N. G. S. N. Y., fifteen shots were fired in 8 seconds actual time. Arrangements have not yet been completed for its manufacture, as patents have just been secured.

ON THE RELATIVE ECONOMY OF VENTILATION BY HEATED CHIMNEYS AND VENTILATION BY FANS.*

BY PROF. W. P. TROWBRIDGE.

OF the various modes of producing the air currents by which vitiated air is removed from chambers, halls, or working places, and fresh or pure air simultaneously introduced, involving the processes of ventilation, the heated chimney is the most common; although it is generally recognized that where large volumes of air are to be moved against considerable passive or frictional resistances, the use of the fan or blower is theoretically the most economical.

The following investigation has been undertaken with a view of establishing the exact theoretical relation between these two modes of ventilation, as far as economy of heat is concerned, and incidentally to determine, as far as the uncertain elements of the question may permit, the circumstances under which either of these methods may advantageously be employed in preference to the other; it being understood that the ventilation of public buildings, mines, suites of rooms, or single large rooms are all included in the problem.

It is assumed that since air is everywhere present at the earth's surface, ordinary ventilation is accomplished by a simple movement of air, all portions of which exist under the same pressure before

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* Read before the American Society of Mechanical Engineers, Chicago Meeting, May, 1886.

motion begins; and that therefore the problem does not involve the lifting of the air through a determined height. The resistance to motion, or the forces to be overcome, are then the frictional resistances of the passages through which the air flows, and the inertia of the air put in motion. The expression frictional resistances is to be understood as implying all those resistances which oppose or obstruct the motions of fluids through conduits or channels, and which are usually expressed in terms of the height due to the actual velocity of flow, or are proportional to the square of the actual velocity of flow.

The work per second necessary to overcome these resistances may be expressed by the weight which flows per second multiplied by the head or height of a column of fluid which, expressed in terms of the velocity of flow, represents the total resistances. The frictional head for a given condition of things—a given conduit and passages—is usually expressed by a constant depending on the length, form and dimensions of the conduit and passages, multiplied by the head due to the velocity of flow; or by an expression having the form

$$F\frac{v^2}{2g}.$$

The work performed in putting a weight of air represented by w in motion with a velocity v per second will be therefore

$$W = w\frac{v^2}{2g}(1 + F).$$

If there are no resistances except those due to the inertia of the moving masses of air the constant quantity F disappears, and the work will be that represented by $w\frac{v^2}{2g}$ — the living force or actual energy imparted to the air per second.

In any investigation having for its object the relative economy of the methods of putting the same quantity of air in movement per second, through the same channels, and with the same velocity, it will be sufficient therefore to consider the work $w\frac{v^2}{2g}$, since the work performed per second in both cases must be the same whether the frictional resistances are considered or not.

It is further to be remarked, that by whatever means air is put

in motion under the circumstances which we are considering, the process consists in a reduction of pressure at one point, whether a fan or a heated chimney be used, which creates an unbalanced head in the surrounding air, and a consequent flow to the point of reduced pressure.

This unbalanced pressure per square foot of section produced by a heated chimney is represented by the expression:

$$(1.) \qquad p = H.(D_a - D_c),$$

in which H represents the height of the chimney, D_a the weight per cubic foot of the external air, and D_c the weight per cubic foot of chimney air.

The height of a column of fluid, whether of chimney air, external air, water, or mercury, which would represent this pressure is found by dividing the above value of p by the density of the fluid; thus,

$$(2.) \qquad \frac{P}{D_c} = H\left(\frac{D_a - D_c}{D_c}\right) = H\left(\frac{T_c - T_a}{T_a}\right), \text{ because } \frac{D_a}{D_c} = \frac{T_c}{T_a}.$$

T_c and T_a representing the absolute temperatures of the chimney air and the external air respectively.

In this last expression $H\left(\frac{T_c - T_a}{T_a}\right)$ represents the height of a column of air of a uniform density D_c, which by its weight would give a pressure per square foot represented by P.

The velocity with which air would flow into a space under this pressure is

$$(3.) \qquad v = \sqrt{2\,gH.\left(\frac{T_c - T_a}{T_a}\right)}.$$

The work per second produced by the chimney for each square foot of cross section will be

$$(4.) \qquad W = pv = D_c\, v.\, H\left(\frac{T_c - T_a}{T_a}\right).$$

Substituting the value of v from (3) we have

$$(5.) \qquad W = D_c\sqrt{2g\,H^3.\left(\frac{T_c - T_a}{T_a}\right)^3} \quad . \quad . \quad . \quad \text{foot lbs.}$$

This is the work per second in foot-pounds accomplished by the expenditure of heat in heating the air of the chimney, and thus producing motion.

The quantity of heat thus expended is represented by the expression

$$Q = D_c\, v.\, c_p\, (T_c - T_a), \tag{6.}$$

in which Q is expressed in units of heat, $\overline{D_c\, v}$ represents the weight of air which passes through each square foot of cross section per second, and $(T_c - T_a)$ the number of degrees through which this air has been heated, and c_p the specific heat of air under constant pressure.

If we substitute again in this expression the value of v taken from (3) we have

$$Q = D_c\, c_p \sqrt{2g\, H \frac{(T_c - T_a)^2}{T_a}}. \tag{7.}$$

This expression represents the heat units expended in heating the air of the chimney to produce the velocity v in the chimney.

The heat furnished may be supplied by a furnace at the base of the chimney, the heated products of combustion from which mingle with the air which enters the base of the chimney; by a system of steam pipes which heat the air by contact as it passes through or among them, or by any other mode which will accomplish the result. If a fire or furnace be employed, as in mines, in such a way that the dissipation or loss of heat from the furnace is prevented, the efficiency of the furnace may be considered unity.

Under this, the most favorable circumstance for the efficiency of the chimney, equation (7) gives the total heat generated and available.

In ventilation by a fan or blower driven by a steam engine, the heat expended to produce the same velocity, or the same discharge and renewal of air, will depend on the efficiency of the steam boiler and engine, the efficiency of the fan or blower, and the loss by friction in the apparatus.

If we consider the efficiency of the boiler and engine to be one-tenth, the efficiency of the fan five-tenths, and the loss from friction two-tenths, or the efficiency as regards friction eight-tenths, the resulting efficiency of the whole apparatus will be

$$E = .1 \times .5 \times .8 = .04 \text{ or } \tfrac{1}{25}.$$

The work performed by the heated chimney to produce the velocity v, and for each square foot of cross section was found to be equation (5)

$$(8.) \qquad W = D_c \sqrt{2g\,H^3 \left(\frac{T_c - T_a}{T_a}\right)^3} \quad . \quad . \text{ in foot lbs.}$$

To produce the same work by a fan whose efficiency is $\frac{1}{25}$, twenty-five times this amount of work must be expended in equivalent heat units.. Hence the number of heat units to be expended will be

$$(9.) \qquad Q^1 = \frac{25}{772} D_c \sqrt{2g\,H^3 \left(\frac{T_c - T_a}{T_a}\right)^3},$$

the second member being divided by 772 to transform its value in foot lbs. to its value in heat units. The relative quantities of heat expended by the chimney and fan, or the relative efficiency under the conditions assumed will then be

$$(10.) \qquad \frac{Q^1}{Q} = \frac{\frac{25}{772} D_c \sqrt{2g\,H^3 \left(\frac{T_c - T_a}{T_a}\right)^3}}{D_c\, c_p \sqrt{2g\,H.\,\frac{(T_c - T_a)^2}{T_a}}}.$$

$$\text{Or} \qquad \frac{Q^1}{Q} = \frac{25\,H}{772.\,c_p\,T_a} = \frac{H}{7.35\,T_a},$$

the value of c_p being 0.238.

If we suppose the temperature of the external air to be 60° F., the value of T_a will be 519.4, and

$$(11.) \qquad \frac{Q^1}{Q} = \frac{H}{3817.59}.$$

This expression shows that the relative efficiency depends only on the height of the chimney, and in no way on the differences of temperatures within and without the chimney. For a chimney one hundred feet high the efficiencies will be as 1 to 38.17; or,

$$Q^1 = \frac{Q}{38.17},$$

showing that the chimney requires an expenditure of heat thirty-eight times greater than the fan. For a chimney 500 feet high, the fan will be 7.6 more efficient.

If the chimney be heated by steam pipes at its base the efficiency of the boiler and pipes must be taken into consideration, making a result still more unfavorable for the chimney.

On the other hand, where small quantities of air are moved, requiring only a fraction of a horse-power, or one or two horse-

powers, to drive a fan, these powers being produced by a small engine and boiler employed solely for this purpose, the efficiency of the mechanical apparatus would probably be much less than $\frac{1}{2}\frac{1}{8}$, a condition of things unfavorable to the fan.

We may now inquire under what circumstances the chimney might be advantageously employed instead of the fan.

In all cases of moderate ventilation of rooms or buildings where as a condition of health or comfort the air must be heated before it enters the rooms, and spontaneous ventilation is produced by the passage of this heated air upwards through vertical flues, the efficiency of this mode of ventilation is evidently unity; that is to say, no special heat is required for ventilation; and *if such ventilation be sufficient*, the process is faultless as far as cost is concerned. This is a condition of things which may be realized in most dwelling-houses, and in many halls, school-rooms, and public buildings, provided inlet and outlet flues of ample cross section be provided, and the heated air be properly distributed.

If, starting from this condition of things, we suppose a more active ventilation to be demanded, but such as requires the smallest amount of power, the cost of this power, when the wages of a skilled mechanic are taken into account, may quite outweigh the advantages of the fan in fuel. There are many cases in which steam pipes in the base of a chimney, requiring absolutely no care or attention, may be preferable to mechanical ventilation, on the ground of cost, and trouble of attendance, repairs, and maintenance. There is quite a wide field for the employment of heated chimneys for ventilation before a limit is reached when the fan becomes indispensable, even when economy alone is considered; and this field becomes more extended, when convenience, saving of time, and personal care and attention influence a choice.

Ventilation by chimneys is disadvantageous under one point of view in any case, viz.: the difficulty of accelerating the ventilation at will when larger quantities of air are needed in emergencies.

The fan or blower possesses the advantage in this respect that by increasing the number of revolutions of the fan the head or pressure is increased, the law being that the total head produced is equal (in centrifugal fans) to twice the height due to the velocity of the extremities of the blades, or

$$H = \frac{v'^2}{g} \text{ approximately in practice.}$$

In mines it is evident that to produce by a chimney the same ventilation as that produced by a fan with the same economy of fuel the up-cast shaft must be very deep. Taking into consideration the wages of an engineer employed to run a large fan and the cost of maintenance and repairs, it might happen, however, that a mine of moderate depth, where the galleries are large and the resistances consequently small, could be efficiently ventilated by a furnace and chimney, at no greater expense than is required for the fan.

It is worth while to consider in this connection the rate at which the expenditure of heat increases in chimney ventilation when for the same channels of flow it is desirable to accelerate the velocity by increasing the heat of the chimney. Equation (3) gives the volume of flow per unit of section of the chimney, in terms of the height of the chimney and the interior and exterior temperatures. For the same height H, the volume of flow per second is proportional to the square root of the difference of temperatures.

Equation (7) gives the expenditure of heat for the same height H, and for the same difference of temperatures. The height H remaining constant, the expenditure of heat is proportional to the square root of the cube of the difference of temperatures.

The first formula is equivalent to the following:

$$V = C\sqrt{T_c - T_a},$$

and the second to

$$Q = C'\sqrt{(T_c - T_a)^3}$$

C and C' being constants.

If in these formulas we make $(T_c - T_a)$ successively 9, 16, 25, 36, 49, 64, 81, we have the following results:

Differences of Temperature.	Volumes.	Heat expended.
9°	C × 3	C′ × 27
16°	C × 4	C′ × 64
25°	C × 5	C′ × 125
36°	C × 6	C′ × 216
49°	C × 7	C′ × 343
64°	C × 8	C′ × 512
81°	C × 9	C′ × 729

This shows that as the volume (or velocity) is increased by increasing the difference of temperature, the expenditure of heat increases as the cubes of the volumes.

Economy of heat requires, therefore, that the *velocity* shall be kept small and increase of *volume* obtained by enlarging the chimney and the channels or conduits through which the air passes. Moreover, since the resistances from friction diminish in rapid proportion as the channels are enlarged, and more of the total head produced by the chimney becomes available to create the velocity of flow, an additional advantage in large cross sections for the chimney and conduits is secured.

The same laws of expenditure of heat hold for the fan or blower, the expenditures of heat increasing for the same conduit as the cube of the velocity of flow. This is, in fact, a general law for all cases where work is performed under such circumstances that the resistances are proportional to the square of the velocity of motion. In such cases the resistance being ($R = C. V^2$) a constant multiplied by the square of the velocity, the work performed per second will be proportional to the cube of the velocity

$$W = Rv = CV^3.$$

It often happens that for a particular chimney and channels of flow the ventilation becomes insufficient, and instead of increasing the heat in the chimney with a large additional expenditure of fuel, a fan is introduced to take the place of the chimney ventilation. The relative efficiency $= Q \frac{Q^1 H}{3817.59}$, and the application of this law of the proportion of heat expended to the velocity of discharge, enables us to ascertain to what limit such a substitution of a fan for a chimney may be carried before the cost of the fan exceeds the cost of the furnace ventilation.

In the above equation of efficiency, if the chimney is 100 feet high the fan will be 38 times more efficient than the chimney, and the table shows that the velocity of flow by the fan may be quadrupled before the cost exceeds that of the chimney. If the chimney is 200 feet high the fan will be 19 times more efficient than the chimney, and the velocity of flow may be increased to a little more than two and a half times that which was produced by the chimney before the cost by the fan exceeds that by the chimney. For a

chimney 500 feet high the velocity by a substituted fan could hardly be made twice that produced by the chimney before the cost of the fan with increased ventilation should exceed that of the chimney. The question might then turn upon the advisability of getting the increase by additional heat in the chimney even with a large proportional additional expenditure of fuel; the cost of attending and maintaining the fan becoming an important element in the problem.

It is quite evident that for the fan as well as for the chimney low velocities and large conduits are favorable to economy.

The following records of experiments are furnished in connection with this paper as a contribution from Mr. George A. Suter, M.E., a graduate of the School of Mines, junior member of the Society, and now engineer for the New York Exhaust Ventilator Company.

RECORD OF EXPERIMENTS MADE WITH THE BLACKMAN FAN BY MR. GEO A. SUTER, M.E., TO DETERMINE THE VOLUMES OF AIR DELIVERED UNDER VARIOUS CONDITIONS, AND THE POWER REQUIRED.

Revolutions per minute.	Cubic feet of air delivered per minute.	Horse-power.	Water-gauge. Inches.	Nature of the experiments.
350	25797	0.65		Drawing air through 30 feet of 48 inch diameter pipe on inlet side of the fan.
440	32575	2.29		
534	41929	4.42		
612	47756	7.41		
340	20372	0.76		Forcing air through 30 feet of 48 inch diameter pipe on outlet side of the fan.
453	26660	1.99		
536	31649	3.86		
627	36543	6.47		
340	9983	1.12	0.28	Drawing air through 30 feet of 48 inch pipe on inlet side of the fan—the pipe being obstructed by a diaphragm of cheese-cloth.
430	13017	3.17	0.47	
534	17018	6.07	0.75	
570	18649	8.46	0.87	
330	8399	1.31	0.26	Forcing air through 30 ft. of 48 inch pipe on outlet side of fan—the pipe being obstructed by a diaphragm of cheese-cloth.
437	10071	3.27	0.45	
516	11157	6.00	0.75	

The experiments were made by him with great care, the power of the engine driving the fan having been determined by the steam-

engine indicator, and the volumes of air delivered having been carefully determined at the same time by an anemometer.

The fan employed was a Blackman fan, belonging to the class of disc fans, four feet diameter, and the experimental apparatus was so arranged that the air was drawn and forced alternately through a metallic tube 30 feet long and 4 feet diameter; the fan being mounted at one end of the tube.

In the first two sets of experiments the only resistance to the flow of the air was the frictional resistance of the tube and fan passages, and the resistance caused by the contracted vein at the entrance. In the second two sets the passage of the air was obstructed by a diaphragm of cheese-cloth placed within the tube; and this additional resistance was ascertained by a water-gauge in the usual way. The table of experiments is useful for determining the horse-power required for given volumes of air discharged with this class of fans, under free delivery and against resistances represented by the water-gauge readings of the table. For large volumes with free delivery, or with very small water-gauges, the efficiency and the small power required are worthy of attention. Estimating four and a half pounds of coal per horse-power per hour in common cases, with coal at five dollars per ton, a horse-power will cost as far as fuel is concerned about one cent per hour; the hire of a man to take care of and manage the apparatus, including other expenses, perhaps twenty-five cents per hour.

For such cases, and especially where the power required is only a *small fraction* of a horse-power as in ventilating single large rooms or small buildings, it is evident that as regards cost of fuel and the care and attention required, ventilation by heated chimneys is to be preferred. Where a fan is driven by machinery employed for other purposes than ventilation, the cost of attendance chargeable to ventilation being therefore trifling, the fan would evidently in all such cases be more appropriate. A variety of circumstances and conditions enter into these problems of ventilating single rooms, or halls, and a choice can only be made through the exercise of the best judgment.

Under circumstances where hospitals or public buildings of considerable magnitude are to be ventilated, and especially where the activity of the ventilation must be varied occasionally, the fan is no doubt to be preferred. And this is quite sure to be the case when the vitiated air is drawn through several systems of collect-

ing ducts from a series of large rooms, into one main outlet; the friction of such collecting conduits, and the resistances of bends and changes of direction in them becoming principal factors in the power consumed. In such cases a comparatively high velocity at the outlet is indispensable. A system of ventilation by means of heated chimneys in such cases involving no greater cost would require numerous and ample vertical heated flues so arranged in the construction of the building that the velocity in each flue should be the smallest possible, and the frictional resistances avoided by the most direct passages of the vitiated air to the heated chimneys. A thorough and proper distribution of the incoming fresh air would demand such a distribution also that collecting ducts could be largely dispensed with; or if they were necessary, that they should be as short and as large in size as the conditions of least resistance might demand.

Such a system might in many cases be preferable to one involving the use of a fan even in large buildings; but unless such buildings have been designed with this plan in view, proper inlets for fresh air forming a part of the plan, it is difficult to apply it with success.

In nearly all public buildings of large size, which come under the head of old buildings in which the necessities of proper ventilation were originally neglected, the fan will probably be found to be the most efficient remedy for deficient ventilation.

THE RIGHT OF LATERAL PURSUIT.

BY WILLARD PARKER BUTLER,

OF THE NEW YORK BAR.

SINCE the March number of the QUARTERLY was published, a decision in mining law has been rendered by the Supreme Court, which is likely to be of some importance, and which has finally appeared in permanent form. This decision was made in the case of the Iron Silver Mining Company against Cheesman (Lime Lode *v.* Smuggler), and is reported in 116 U. S. Reports, page 529. This case, as far as the Colorado Circuit is concerned, will doubt-

less very much limit the law of the apex, for in view of the fact that the charge of Judge Hallett on the trial of the case at circuit, has been distinctly affirmed and highly commended by the Supreme Court, probably no judge will be found, at least in the territory included in that circuit, who will feel at liberty to extend the limits defined by Judge Hallett in his charge, and no jury will be willing to find for a plaintiff on any such evidence as that offered in the Smuggler case, which has been twice decided in favor of the defendants.

In the opinion of the Supreme Court, Mr. Justice Miller, who writes it, says, quoting from Judge Hallett's charge: "The proposition of the plaintiff is that the evidence before you (the jury) shows that a lode exists in the ground in controversy as already defined. The defendants deny that proposition, and the case turns on that question. They concede that there is, in the territory opened by the works, ore in detached masses or fragments, but so intermingled with the enclosing rocks that it cannot be regarded as a continuous body, or as marking the line of a lode or vein. All that has been said by witnesses about rock in place is valuable only as it tends to prove or disprove the existence of a crevice or opening extending from one claim to the other. Excluding the wash, slide, or débris on the surface of the mountain, all things in the mass of the mountain are in place. A continuous body of mineral or mineral-bearing rock, extending through loose and disjointed rocks, is a lode as fully as that which is found in more regular formation, but if it is not continuous, or is not found in a crevice or opening which is itself continuous, it cannot be called by that name. In that case it lacks the individuality and extension which is an essential quality of a lode or vein. Recognizing this, the plaintiff has given evidence to establish the existence of porphyry and lime in regular order with an opening between them, filled with vein matter."

The defendants sought to show that the ground was broken and disjointed, and the several parts so intermingled that no lode could extend from one claim to the other.

Upon this state of facts and charge, the jury found for the defendants, namely, that there was no lode, and that therefore the possession of the apex by the plaintiff made no difference, and that the plaintiffs were not entitled to pursue the vein outside of their lateral lines, as far as the vein extended.

This case assumes that the plaintiffs were the actual possessors

of what the statute terms the "top or apex" of the lode. The decision turns purely upon the question—was or was not the vein worked by the defendants a continuation of that whose apex was possessed by the plaintiff?

Continuity of the body of mineral or mineral-bearing rock, is again here made the test of the latter being a vein or lode within the meaning of the statute, for the Supreme Court says, p. 537, following Judge Hallett: "If it is not continuous, or is not found in a crevice or opening which is itself continuous, it cannot be called by that name (*i.e.*, of vein or lode)." The Court also cites Mr. Justice Field's opinion in the Eureka case, 4 Sawyer, 311, to the effect that "A continuous body of mineralized rock lying within any other well-defined boundaries, on the earth's surface and under it, would equally constitute a lode." The Court does not attempt to define what is or is not continuity, to any further extent than has been already done by Judge Hallett, and whether the body of mineral or mineral-bearing rock, crevice, or opening is or is not strictly continuous, is still a question for the jury. The cases are so rare where continuity can be strictly shown, that undoubtedly the only feasible way for the jury will be, in the majority of cases, to find that there is no continuity and hence no vein or lode. This, in my opinion, must be construed to limit practically the right of lateral pursuit to cases where the continuity of the mineral-bearing rock, crevice, or opening is notoriously a matter of ocular perception.

The Supreme Court seems to me to have shirked the responsibility of deciding what, in its opinion, are the elements which go to show such continuity as is necessary for the deposit or vein to be a lode under the statute, except to the extent already determined by Judge Hallett, and while it will greatly simplify matters to have juries find "no continuity," in the great majority of cases, it is easy to see how, in many cases, great injustice may be done, for the reason that no jury when instructed in what the Supreme Court has decided in a proper manner, will be willing to assume a responsibility which the Supreme Court has apparently declined to shoulder.

THE POINT SHIRLEY COPPER WORKS.

BY T. EGLESTON, PH.D.

THE Revere Copper Company's smelting works were constructed in the year 1844, at Point Shirley, a promontory in Boston harbor, about five miles from the city, to treat foreign ores that were brought to the United States. They comprised an area of five or six acres, exposed to the water on three sides, a peculiarly favorable location, as the sulphurous smoke was blown out to sea, and there were no damages to pay from this cause. The works had a wharf at which vessels of the largest size could come to load or unload. The bulk of the ore treated came from South America, and were called Chili ores, whether they came from that country or not. These were collected largely at Valparaiso. Some of them came from Peru and other South American States. They were purchased as return freight when they could be had at a very low price, by captains of vessels in the South American trade, and brought home generally in ballast, but sometimes as cargo. They were usually rich oxides, sulphides, phosphates, arsenides, silicates, and carbonates from Chili. A few ores were brought in from Vershire, Vermont, distant about 60 miles by rail, while the freights were low. For a single year the products of the Cliff mine, when it was first opened, were refined at Canton where their refining works are now situated; those of the Isle Royale, and of the Pewabic mines, were, for several years, treated at Point Shirley. The American ores, however, were but a small portion of those treated, never over ten per cent., and in later years none at all. As long as the mines in Cuba lasted the bulk of their product was treated, and, in addition, very rich malachites were received from Africa.

These ores came at very irregular intervals, but there were always enough on hand for continuous treatment, but all of the furnaces were not always in blast at the same time. The ores were mostly sulphides, and did not contain either silver or gold; most of them were pure, but when a small quantity of very rich impure ores, such as carbonates resulting from the decomposition of tetrahedrite, and consequently, containing considerable arsenic and

antimony, could be had at a very low price, these were purchased and added to the heaps, in small quantities at a time, with the others, though they were always roasted separately when in large quantity. The ores, though they were for the most part sulphides, were of every variety and richness. Small quantities of oxides were brought from both Cuba and South America, with some arsenides and silicates. Tetrahedrite was occasionally purchased when sufficient iron pyrites, with or without copper, could be had to justify its treatment, but generally but small quantities of it were purchased at any price or time. The ores at first were very rich, but grew poorer as they commenced to be more in demand. At first they averaged 25 per cent., then 16 per cent. When England took off the duty on copper-ore and the United States put it on they were averaging about 10 per cent. When the duty was put on in 1869 the works ran a few years on what ores they could get in the United States, and this was a very uncertain quantity. They finally closed down in the year 1872. The largest amount treated in any one year was 25,000 tons of 20 per cent. ore, yielding, consequently, 5000 tons of refined copper. They averaged from 15,000 to 20,000 tons a year. At the time these works were built the commercial conditions were peculiarly favorable. England had imposed a duty of 4 per cent. on all copper-ores entering her ports. The United States admitted them free. The works could, therefore, on account of their very advantageous situation, easily compete with the cheap coal and labor of the English works. At the time the works were closed the situation was reversed. England had taken off all duty on copper-ores, and the United States had put one on for the purpose of protecting the Lake Superior copper mines. As competition under such conditions was out of the question the works finally closed, legislated out of existence.

The ores which were treated may be divided into four categories:

Pure sulphurous ores.

Rich impure sulphurous ores.

Impure oxidized ores.

Pure and very rich oxides.

From some points of view these works were very interesting. They were among the first American copper works using the continental processes, successfully run in this country. They are an important step in the history of copper metallurgy in the United States. Their technical success was largely owing, in the first

instance, to the ability of the chemists of the works. The first one was Frederick W. Davis, who died in 1854. He was succeeded by George J. Dickinson, who died in 1879. Both these gentlemen were men of great ability, and in their day had few equals, and certainly no superiors.

As I had occasion to visit Point Shirley while the works were running in the greatest prosperity, and as no description of them has been published except in French,* I have written out the notes which I made at the time, and added such information as I have been able to obtain from the company, in order that they may be accessible to those who may wish to consult them.

Point Shirley is an almost desert promontory. Vessels came laden with ores and material to the wharves, and owing to the situation of the works there were no expenses arising from damages done by the escaping gases, as has been stated. The works were surrounded on three sides by a board fence about five feet high. On the land side and wherever else it was necessary to protect the roasting piles from the wind it was made of closed boards. Elsewhere it was of cedar poles nailed to posts as slats, and, consequently, quite open. Railroads were used all through the works for transportation. As soon as the vessels arrived at the dock the ores were discharged and weighed on the spot. They were then carried according to their character and richness to the place for the storage of the different ores. The general disposition of the works was such, as to have the raw ore go to one side of the works, in one direction, and to return the copper to the store houses, near the wharves in the opposite direction, without having any of the lines of transportation cross each other.

The prices paid for the different materials used is given below. The cost of treatment was very high:

Charcoal per bushel of 20 pounds,	$0 10 to 0 17
Cord wood, per cord,	5 00 to 7 00
Anthracite,	6 00 to 9 00
Day labor,	1 25
Skilled labor,	1 75 to 2 00
Superintendent,	3 00 to 3 50

The impure carbonates and the sulphurous ores were roasted in large piles in the open air. They were then melted in a shaft

* Rivot, Traitment des Minerais Metalliques, vol. i., p. 397, Paris, 1860.

furnace; in this fusion, when pure ores were treated, the rich and pure scorias were added. A scoria was produced which was for the most part sufficiently poor to be thrown away. The small quantity of rich slags produced were re-treated. The matte contained 40 to 45 per cent. of copper; sometimes some black copper was also produced, but this was avoided as much as possible, as it was likely to be impure. The matte was roasted in stalls of a peculiar construction with two fires. The roasted mattes were then melted in a shaft furnace with rich and pure oxides and calcined carbonates. This fusion gave black copper, containing 90 to 91 per cent. of copper, a small quantity of rich mattes, and a scoria which was remelted in the fusion for mattes. The black copper was fined and refined in a special works, at Canton, where the rolling-mills were and are, which was not a good disposition, since the black copper had to be transported there and the scoria brought back. The fining and refining were done in a reverberatory furnace. The process as a whole, therefore, consisted of five operations:

1. Roasting the ore in piles.
2. Fusion of the roasted ore for mattes in a shaft furnace.
3. Roasting the mattes in the Boston stalls.
4. Fusion for black copper in a shaft furnace.
5. Fining and refining in a reverberatory furnace.

The roasting heaps were situated near the docks in the open air. The ores were classified and kept under sheds behind the smelting works, properly speaking, but in front of the ore heaps. The shaft furnaces were in a long building with large doors, to allow of free exit and entry. There were twelve shaft furnaces all in a row in one building. They were all alike. Behind the shaft furnaces were two tracks for the transportation of the scorias and ore. On the other side of these tracks were twenty-four roasting kilns under sheds, built in rows of eight to twelve each, back to back, with a wide passage way in front. There were four reverberatory furnaces at Canton for fining and refining, two of which were in use at once.

When the works were first built only four of these processes were carried out at Point Shirley. Just before the works closed four refining furnaces were put up at Point Shirley, and four

large calcining furnaces for roasting the rich mattes, but they were only used for a short time, though during that time the work of refining was conducted there and at Canton as well.

The ores as they arrived at the works were assayed not only for the quantity of copper they contained, but also for the nature of their gangues and their impurities, especially arsenic and antimony. All the ores which contained these, whether carbonates or not, it was necessary to roast, but these ores could not have been roasted unless there was a large quantity of sulphur present in the ores roasted with them, for which purpose sulphurous ores poor in copper and rich in sulphur, or even iron pyrites, were purchased to add to them. They had to be mixed as intimately as possible with the pulverized sulphurous ores. If this was not done no very appreciable quantity of arsenic and antimony would be driven off during the roasting. The ore was generally roasted as soon as it arrived, so that the quantity to be roasted depended on the number of cargoes arriving at any given time.

I. Roasting the Ore.

The ore was roasted in rectangular piles of various sizes, depending on the supply of ore, which was very irregular. When the ores were plenty the piles contained from 100 to 250 tons; when there was but little they were sometimes as small as 50 tons. All the pure ores were roasted separately. Those containing arsenic and antimony, especially the carbonates and other products resulting from the decomposition of tetrahedrite, were treated by themselves, as they gave an impure copper. The piles of such ores were always small. They were always treated with pyrites in order to have an excess of sulphur. Occasionally these pyrites contained a small quantity of copper, too small, however, to justify its treatment by itself. Generally, however, the pyrites contained no copper. Usually the piles did not contain less than 100 tons. This was about the average of them all. No time was lost in roasting the ores. As soon as their quality was ascertained, and there was a roasting space vacant, they were sent without delay to be roasted. The piles were always made on areas specially prepared for the purpose. The base was made of clay, well beaten, so as to make the ground impervious to air. They were raised a few degrees in the center. After a certain number of months the

hearth became impregnated with the copper mostly as oxide. It was dug up, its value determined, and was used in small quantities as a flux, in the shaft furnace operations when it was needed. The amount of material thus collected was from 800 to 1000 tons a year. It contained from five to six per cent. of copper. The base of the pile was made of two layers of wood. The first one was laid perpendicular to the length of the pile, each piece was about six feet long, the space between each of these being from four to six inches. The upper layer was smaller wood, and was laid at right angles to the first. The ore was placed on this, the large pieces being placed first, the smaller ones next, and so on. Two or three chimneys built of large pieces of ore were left in the length of the pile. The whole was covered with four to six inches of fine ore. The cover rested on the first layer of large ore. The whole pile was between six and seven feet high. Shavings or small wood were placed between the first range of wood. These were fired, and the combustion was made to run through the pile by means of the chimneys. When the whole was lighted the chimneys were filled with ore, and any damage that might have been done to the cover by the sinking of the pile owing to the burning out of the wood was repaired. The length of time required to roast a pile of ore varied with the season, as the piles were not covered. If the winds were not very high the roasting could have been finished at the end of one month. The roasting in such a case would have been incomplete, because the temperature had been sufficiently high to melt some of the ore which was not then roasted. Sometimes violent rains put the pile out; it had then to be remade. The making of such a pile would take from one to two weeks. It would take from a week to ten days to run the whole of it through the furnace. The roasting would take about eight weeks, depending somewhat on the season of the year, so that each pile would occupy the ground for about three months. The smaller piles were from three to six weeks in roasting, and did not occupy the ground more than two months. To protect the piles against violent winds, a fence five feet high was erected towards both the land and the bay. The roasting ground was too large to be covered with a roof. A small quantity of sulphur condensed on the outside of these piles, but no attempt was made to render the condensation perfect, or to collect the sulphur, since it was likely to contain arsenic and antimony, and in any case

would hardly have paid the expenses of collecting it. All of this sulphur was subsequently volatilized by the heat of the pile. When the roasting was completed, and the pile cooled, the ore was picked, and the pieces which had been incompletely roasted were treated again. The quantity to be treated a second time depended on the weather. It usually required five cords of wood for a pile of 100 tons of good ore. The impure ores took somewhat less on account of the large quantity of iron pyrites added to the ore, which furnished considerable heat from the burning of the sulphur. A little more fuel was required in winter than in summer, but five cords was a fair average. About 20,000 tons of sulphurous ores had to be roasted in the year. This would require 200 piles of 100 tons each to be made, so that an area for 50 to 60 piles of 100 tons each had to be provided. If the piles were very small a single chimney only was made in them. If the piles were very large three, or sometimes more, chimneys, depending on the character of the ore, were made. Six men were kept for the work of making the piles. They were aided in their work by four or five laborers, the number depending on the supply of ore and the press of the work. These men were paid like all the others in the works, for common labor $1.25, and for skilled labor $1.75 to $2 a day. The labor was entirely confined to making the piles and carrying away the roasted ore. The pile once made required little looking after.

The expenses of roasting for one ton of ore were:

0.05 cords of wood, at $3.50,	$0.175
Labor, 0.15 days,	0.187
	$0.362

The men who took care of the piles had only to watch them and repair the cover when necessary. The ore was brought by the furnace laborers, and the roasted ore carried away by them.

II. Fusion for Mattes.

The shaft furnaces which were used were rectangular in shape and quite low, constructed with a view to using pure ores only. They were too low for impure, but quite suitable for pure ores. Whatever rich impure ores were to be treated, it was necessary to

add in very small quantities with the pure ores, which was a very bad plan, or else they were treated entirely by themselves, in which case the furnaces were too low to give anything but a very impure copper. If there had been much impure ore it would have been very easy to have kept one furnace working on them, arranged at such a height that all the impure ores treated in it would have remained a very long time at a low temperature with a feeble reducing action, which is the condition most favorable for the reduction and volatilization of arsenic and antimony. The dimensions usually adopted are given below:

	Feet.	Inches.
Total height of furnace and chimney,	33	
Tuyere to throat,	5	
Heighth of tuyere above the bottom of the crucible,	1	4
Section of the furnace at the tuyeres' inside measure,	2 × 3	
Depth of the breast below the front hearth,		6
Size of the charging door,	3 × 3	
Front hearth extends beyond the breast,		6
Diameter of tuyeres,		1½
Thickness of fire-brick,		11
Thickness of common brick,		18
Distance between common and fire-brick,		½
Length of cast-iron side in front of the furnace,	3	10
Depth and diameter of slag basin,		18

There were twelve of these furnaces in a line, two of which were used on black copper. The construction of the shaft furnace was different at different periods of the works. At first the charging-door was in front, directly over the tap-hole. To make the charge the men got up on the apron of the front hearth with baskets of the charge on their heads, and threw the ore into the furnace by the charging-door. To prevent the steep of the front hearth from becoming damaged it was covered with a sheet of wrought iron during this operation. As there was no provision to see into the furnace, the necessity of charging was judged of by the brightness of the flame at the throat. It was quite difficult to keep the charges in their proper condition in this way, and, subsequently, this was modified so as to have the charging-floor over the tuyere, as is usual in such furnaces. In front of the furnace cast-iron plates were placed, making a box the width of the furnace and 3 feet 10 inches long. The space so made was filled to the level of the tuyere with a steep, made at first of six parts of ground burned clay, six of powdered charcoal, and eighteen of ground coke. Towards the last part of the time anthracite dust was used in

place of the coke. The anthracite was purchased especially for this purpose. Sometimes the steep was made of six parts of burned clay and twenty-four of anthracite dust. Both the charcoal and the anthracite were refuse material. They were pulverized together in stamps and then screened, and moistened with a little water. This steep was put in in small quantities at a time and pounded down, the surface of each successive bed being deeply scraped with a point, so as to be sure that when heated there should be no solution of continuity. When the whole was full the crucible was cut out. To form the tap-hole a stick of wood was put in as soon as the steep reached the proper height, which was withdrawn when the crucible of the furnace was cut out. At first the front of the cast-iron box was close to the breast, only space enough being left for the work of the furnace. Afterwards it was made long, the surface of the top being slightly inclined, and towards its end a basin was cut out in it, into which the slags were made to flow, so that before becoming solid they could deposit there any copper granules they held in suspension. The matte was tapped in a basin lined with steep in the floor of the works, and was taken off in rosettes.

The breast of the furnace was made only one brick thick, which gave complete control of the furnace when engorgements occurred, as it was only necessary to take out the bricks where the engorgement was and remove it. A space was left between the ordinary and refractory masonry, filled with broken brick to allow of easy repair, and also for contraction and expansion. On the tuyere side of the furnace on the level of the charging floor there was an incline of the masonry to prevent the ore resting there. The opening for charging was on the tuyere side of the furnace, and was six feet from the ground. The furnaces were blown with D-shaped, water tuyeres, with an opening equivalent to three inches in diameter. At first these were made of cast-iron like those used in an ordinary blast furnace, but they were subsequently made of tubes of composition metal cast at the works and sweated together. The nozzle did not fill the eye of the tuyere completely. The roasted ores which still contained arsenic and antimony, if they were in the ore, were treated in very small quantities at a time in this furnace. The effort was always made so to associate the ores, that the gangues would flux themselves. When there was an excess of silica, however, a flux was added. Oyster shells were ordinarily used since

these could be had cheap. A very considerable portion of scorias from the black copper fusion were also added, together with débris of the refining furnaces. The scorias from a preceding operation containing black copper were not used in this charge, but in the succeeding fusion. There was generally one of scoria for two of ore, in the charge but sometimes as much scoria as ore was added. The charge was arranged on the platform at the level of the throat. It was generally made in a rectangular heap, in which there was a layer of roasted and oxidized ores, then flux, and above scoria. Two stratifications arranged in this way were generally made for each charge. The charge was thus made up of six beds. The pile so arranged was generally about eighteen inches high. The workman was expected to take a shovelful of the entire height of the charge so arranged, which was thrown into a basket to be carried to the furnace. This charge was about the same during the entire blast. The furnaces were blown with a double cylinder Tuft's beam-blast engine, moved by a horizontal steam-engine. A Sturtevant's fan blower was, besides, kept in reserve. The pressure of blast used for the furnace was 1½ pounds.

The fusion bed while the charging-door was over the breast, was made on the floor of the works, and was always in the way of the men. As soon as this was changed, it was made on the charging-floor proper, generally to the right of the charging-door. Each pile contained from three to five tons of the charge. It varied in composition according to the kind of ore to be treated. The composition of the bed was made taking the pure roasted ore as unity. The flux which was used was oyster shells. It varied in quantity with the nature of the ore treated, from 10 to 15 per cent. To this charge were added, in small quantities, any impure ores, whether oxides or roasted, and any material around the works that contained copper, such as the hearths of the furnaces and of the roasting piles, and the rich slags from the black copper fusion, and sometimes the scorias from the first part of the operation of fining. In general the bed was made up of:

Roasted ore,	100
Black copper and fining scorias,	33
Oyster shells,	10
Roasting and furnace hearths, and débris of furnaces,	5
Impure ores,	3
	151

This was made in layers, and was charged in baskets, measured but not weighed. The coal was charged in the same way. The proportion varied from time to time with the nature of the ore, and also with the condition of the furnace, according as the slags varied in their character. These slags were assayed to ascertain their copper contents, but were not analyzed. Their character was judged of by the eye.

When the furnace had been repaired it was allowed to remain for a day or two to dry. A small charcoal fire was then made in it, which was gradually increased until the furnace was half full of charcoal, at which time it was at a red heat. Then anthracite was charged and the blast turned on. As soon as this was done scorias were charged on the tuyere side to form the nose or prolongation of the tuyere in slag. This was done in from twenty-four to thirty-six hours from the start. When the nose was made the charge was introduced against the tuyere side, and the anthracite on the breast side. It required considerable dexterity to charge and keep the anthracite against the breast of the furnace; when the fuel got into the column of ore the nose was very soon destroyed. At first the furnace was blown at a low pressure. This was gradually increased until it arrived at its maximum. This was in about three days, after which time the workings of the furnace were modified from time to time to suit the varying and accidental conditions of the working. These were watched by the overseer, who indicated to the man in charge what changes were to be made. A campaign or blast lasted generally for about fourteen days. By that time the sides of the furnace were very much worn, and it was necessary to go out of blast. During this time from five to six tons of ore per day had been put through, the smaller amount at the commencement, and the larger at the end of the campaign. The blast was brought to a close by the fire-brick near the tuyere being cut out by the iron of the charge. The quantity treated in a single day depended on the character of the ore, and was regulated to produce clean slags as much as possible, rather than to pass a large quantity of ore. The blasts were always regulated so as to terminate on Saturday afternoon, so as to give the furnace time to cool down in the night, so that repairs might be made on Sunday. The repairs did not usually take more than a single day. During this time, if no accident happened to the furnace, nine tons of the charge on an

average will have passed. In the last days of the blast this often amounted to ten or even eleven tons in twenty-four hours, but the furnace at this time was very much enlarged, and but for the corrosion this production might continue. The quantity passed when the furnace was in good working order was very large. The working was excellent, both as to quality and quantity, when the ores were pure, but the furnace and process would not have worked on impure ores alone. The only difficult work was keeping the nose in good order, which could only be done by the greatest watchfulness on the part of the workman, and from it he got many of the indications of working. If it was too red there was danger of its melting off, and the furnace was too hot. If it was not red then the furnace was working too cold. It was carefully watched, for while it was not easy to form the nose at the commencement of the blast, it was exceedingly difficult to repair it when it became damaged. It was necessary to have it long enough to pass through the vertical column of the charge. If shorter, there would be danger of the iron in the ore becoming reduced.

The temperature of the furnace was very high, and all the gases which escaped at the tympe burned. The carbonic oxide at the throat did not, however, burn. As the temperature was very high in the furnace, but little arsenic and antimony was separated, it was in the roasting that this was done, if at all. Ores containing a very large percentage of these substances could not be treated with anthracite in such low furnaces. In the hearth there were mattes and scoria. The slag was allowed to flow off when it rose up to the cinder notch. This opening was kept closed by a large piece of scoria, which the workman removed from time to time, or was allowed to flow continuously. When the matte rose to the level of the breast it was cast by opening the tap-hole. It was collected in an outside basin made of brasque, which was previously dried with hot scoria. A small quantity of scoria always ran out with the matte. This was taken off and always carried with it a certain proportion of matte. This scoria was put back into the charge, as it was very rich in copper The charge was made to contain a large amount of silica, so as to make the slags poor in copper, and retain at the high temperature the most of the oxide of iron in the slag, and thus keep the iron from being reduced. They ran continuously during the whole of the campaign. The color and fluidity gave to the men the in-

dications of the change to be made in the charge. They contained 50 to 52 per cent. of silica. This is more silica than a bisilicate. This should be so, for anthracite requires a high-pressure combustion. The temperature was so high that if silica were not in excess deposits of iron would be formed. As soon as the matte rose so near the tuyere that the scorias commenced to contain grains of matte the casting was made. This was three or four times during a shift, or every three to four hours. Previous to doing this the basin into which the matte was tapped was repaired with steep and dried with the red-hot slag. The tap-hole was pierced with a pointed iron rod. The blast was stopped, and the contents of the crucible of the furnace discharged into the reception basin. Generally, in order to prevent any of the charge sticking to the bottom of the furnace, the tap-hole was closed with a conical piece of clay on the end of a rod, as soon as the slag began to appear. It was not done quickly enough to prevent some of the slag coming out with the matte. This was easily taken off when the matte cooled, by a stroke of the hammer, and was put into the charging-bed with the rest of the rich scorias in the fusion for black copper. The blast was turned on again as quickly as possible after the matte was all out of the furnace, for if it had been turned off for any length of time there would have been danger of the nose melting which could only be kept in place by the cooling influence of the blast.

Before closing the tap-hole it was cleaned from any matters adhering to it. Before turning on the blast the fore-hearth and the crucible of the furnace were cleaned if there was any occasion for it. The fore-hearth was then closed with pieces of slag and fine anthracite. All this work was done very rapidly, as a very little delay would have made trouble, first, by endangering the nose; second, by cooling the lower part of the furnace unnecessarily. The only work that was difficult was that of working the interior of the furnace to remove any obstructions, and the beginning of any engorgement. After the slag had been removed the matte was cast. This was at first done into a reception basin from which it was removed in rosettes, but toward the last was done in a sand bed. The rosettes were made from an inch to an inch and a half thick. The matte was weighed, broken up with a hammer by the men who wheel the ore, and carried by them to the roasting stalls.

The furnaces were too low to expect to find any great difference

between the purity of the roasted ores and the mattes. Most of the impurities contained in the roasted ore was in the mattes. They amounted to about 56 per cent. of the weight of the ore. They contained 34 to 35 per cent. of copper. They were cast in thin plates or rosettes, and it was necessary to break them up before roasting, Their usual composition was approximately:

Copper,	30 to 35
Iron,	40 to 37
Sulphur,	30 to 28

besides small quantities of lead or zinc if there was any in the ore. At the bottom of the casting-basin a certain amount of black copper was often found. This black copper was impure, and every effort was made to prevent its being formed. It was difficult to prevent it, however, for the ores were rich in copper and poor in sulphur. The poor scorias were carried by railroad to the bay, and were used to make ground.

The number of furnaces at work at any given time depended on the quantity of ore; it was generally not less than eight out of the twelve devoted to the fusion. They varied in size, the first being capable of 5 to 6 tons of roasted ore per day. Those constructed toward the last were much larger, and could treat 8 to 10 tons of ore. For each one of these furnaces there were two men per shift of twelve hours, one of them being especially charged with the conduct of the furnace, the other a helper. Besides these there were, for all the furnaces, an overseer, who generally worked only during the day; he had an assistant who took charge of the furnaces at night; and there were four to six laborers who worked only during the day, bringing in the ore and carrying away the slag.

The labor for the whole furnace was:

1 overseer, paid per day,	$2 50
1 overseer's assistant, paid per day,	1 75
4 laborers, paid per day,	1 00
14 to 16 furnace men, paid per day,	1 75
14 to 16 furnace men, assistants, paid per day,	1 25

Expenses for a campaign of fifteen days:

Amount of ore treated, 90 roasted, 5 rich oxidized ores.	95 tons.
Lighting the furnaces,	$6 00
Anthracite, 16 tons, at $8.40,	134 44
Charcoal for reception-basin,	12 00
Founder, sixty days at $1.75,	105 00
Assistants, five days at $1.25,	6 25
Repairs,	11 00
Cost of treating 95 tons of ore,	$274 69
Cost of 1 ton of ore,	2 785

The products obtained were:

Mattes of 45 per cent. of copper,	25 tons
Black copper of 85 per cent.,	5 tons

III. Roasting the Mattes.

The mattes were roasted in stalls or kilns, which varied in dimensions from time to time. These consisted of an arched inclosure. The dimensions varied considerably as they were not built from drawings and were only approximatively alike. The height varied from 5 feet to 6 feet 6 inches. The width from 5 to 6 feet. The depth from 6 feet to 8 feet 3 inches. The side walls between two kilns were 9 inches, and the brick wall 18 inches thick. They were open in front, with a grate, fixed 18 inches to 2 feet from the ground, upon which the fuel and matte to be roasted were placed. In the last year of the works the dimensions were: height 5 to 6 feet, depth 6 feet. The ash-pit was closed by a door, which allowed of regulating the draft perfectly. Just under the arch was an opening to conduct the escaping gases into an arched horizontal flue, 0.45 inch wide, supported on arches built outside of the furnace, placed at regular intervals behind the stalls, which conducted the gases to the chimney and served all the stalls of the works. There were twenty-four of these kilns arranged under the sheds, with a chimney 30 feet high for each set of eight.

The matte was brought to the stall by the laborers, broken into pieces of nearly uniform size. They were about three inches square, and from an inch to an inch and a half thick. There was some variation in the size of the pieces, but few of them were very small. They were, however, separated for the charging, the largest pieces being put on the grate, and the others above it. There were not more than 10 to 15 per cent. of fines made in breaking

the matte up to charge the stall. The front of the stall was made of bricks laid up dry. It was taken down every time a stall was to be discharged, and put up again when it was recharged. The draft was regulated by opening the door of the ash-pit more or less wide. This had the very great advantage of perfectly regulating the draft so that the roasting was very uniformly done.

When the kiln was to be charged a small quantity of wood and a little charcoal was placed on the grate. Above, the matte was charged, putting the large pieces first, and covering the whole with fine ore. There was never more than five feet of charge above the grate. The upper part did not reach to the flue which carried off the gas. About 6 tons of matte were charged in each kiln for the first roasting. The bricks in the front were then laid up, leaving some openings for the passage of air. The fire was lighted by placing shavings underneath the grate. All that the workmen had to do was to see that the fire was well lighted as far as possible over the whole surface of the grate at the same time; and that the door of the ash-pit was sufficiently open to admit the air necessary for combustion, but not enough to raise the temperature so high as to be in danger of fusing the matte. The ash-pit door and the dry brick wall gave entire control of the heat, and it was rarely that any of the matte was fused.

This construction of the stall with an arched roof, and complete control of the air which entered beneath the grate, as well as the amount of fuel to be used, caused the roasting to be done in a very uniform manner. The reactions which took place were more regular and quicker, so that instead of the large number of fires required as in the open stalls, even for moderately pure ores, only two were required here, and not more than three for ores which were quite impure. The heat of the burning wood was quite sufficient to carry the temperature up to the point where the sulphur being driven off, the metals were oxidized. After the wood was burned out the combustion of the sulphur continued, the heat of its burning being great enough to carry on the reactions. No fuel was added in the mattes above the grate, as is necessary when the kiln is not covered. The only precaution necessary to be observed was that there should not be too much fines charged, and that what were charged should be scattered through the pile in such a way as not to obstruct the draft and prevent the currents of air from acting on the larger pieces as they should. Care was taken not

to make the larger pieces of too great size, for then they would have been affected on the outside only, and too large a proportion would have had to be passed in the next fire. Just to what size it was best to break the mattes depended on how compact they were, and how much foreign metals, especially zinc, they contained. Most of the large-sized pieces had to be passed through another fire. In the second fire a little more wood was used, as part of the sulphur had been driven out of the matte, and it was a little less combustible. The operation was, however, just as easily conducted as before, as the whole of it was completely under control, and so much of the sulphur was driven off, that in the next fusion but small quantities of matte were formed. As the heat was not high and the air was in excess, the sulphur, and any arsenic and antimony, if there was any, were volatilized. The heat was not sufficiently high for the impurities to form any stable compounds, and consequently none were formed. The only places where there would have been any danger, was next the wood fire when it was first lighted, and in the fines if there were any there, but these were not put in in that position, and little or no fusion took place in the large pieces. Each operation lasted three days, by which time the fire had reached the fines on top. The front wall was then taken down, and the contents discharged. That sufficiently roasted was sent to the furnace. This was about one-third of the charge. The rest had to be roasted in the same or a neighboring stall. The number of times the matte had to be roasted depended on the purity of the ore. For pure ores it was treated twice, for impure ones three times. When the ores contained considerable blende they all had to be treated with three fires. The reason for this was that the mattes from such ores were quite compact, and that the first fires did not drive off the zinc and the sulphur in sufficient quantities to make it porous enough to be treated in the shaft furnace. When the ores contained arsenic and antimony the object of the third roasting was to get rid of them more effectually. At each successive roasting the quantity treated was a little less than six tons. Each roasting required a little more wood than the previous one as the matte contained less sulphur, and some small quantity of fuel was sometimes introduced in the matte to insure complete roasting. After the second roasting the matte from pure ores contained about ten per cent. of sulphur. It took five days to roast six tons of matte. The number of stalls

varied at different times. They were arranged in two parallel buildings, containing each from eight to twelve stalls, so that there were from sixteen to twenty-four stalls in the works. In each building there were two men to take care of the stalls, who were paid $1.75 a day. They had two men to help them, at $1.25 a day. The stalls thus had eight men. They arranged the charge in the stalls but did not transport the matte. The workmen connected with the shaft furnace brought the mattes and carried the roasted matte away. Each kiln contained six tons; the expense for this amount are:

Wood for lighting,	0.662
Labor, 2.25 at $1.25,	2.817
Charcoal for lighting,	0.100
Cost for 6 tons of mattes,	3.579
Cost for 1 ton of mattes,	0.596

For 1 ton of ore:

Wood, 0.0148 cord,	0.0566
Labor, 0.125,	0.1312
Cost of roasting for 1 ton of ore,	0.1878

The roasted matte was melted for black copper.

IV. Fusion for Black Copper.

In the fusion for black copper the roasted matte and rich oxidized ores, free from arsenic and antimony, were treated. As fluxes the silicious scorias of the first fusion for mattes, old furnace hearths, debris of furnaces, the rich parts of the bottoms of the roasting piles, and any other silicious materials containing copper made in the works where pure ores have been roasted, were used. Such materials made in the treatment of impure ores were all passed in the fusion for mattes, as in this fusion and subsequent roasting of the matte only could the impurities have been separated. If passed here these impurities would all have gone into the black copper, and would have given a very impure product. These materials were added in the fusion bed in such proportions, as were required both to regain the copper they contained, and to make fusible slag. All the greater quantity of silicious material to scorify the iron was passed in this fusion, as the rich oxides contained less quartz.

If the gangues were only calcareous the quantity of scoria was maximum. To these materials were added the small quantities of very rich scorias produced in refining, and all the rich scorias from the fusion for mattes, and from this fusion also, that containing any considerable quantity of the mattes or black copper in grains, such as the scorias that lay next the mattes, or the black copper at the time of casting. The composition of the charge was varied according to the indications given by the scorias, but only such quantities of silicious materials as were necessary to form the slag were put in. The charge and the work in general were conducted as in the fusion for mattes. The furnaces used had the same dimensions as those used in the fusion for mattes. They could not be higher, as the black copper contained so much iron that there would have been danger of its being reduced, and that would both have caused trouble in the working and have made rich sows to be treated. The object of the fusion was to reduce nearly the whole of the oxide of copper contained in the roasted matte to metallic copper so rapidly, that it would be accomplished before the heat was sufficient for the silica to attack the oxides, and also before the iron had time to become reduced. This was aided by the reaction which took place between the sulphide of iron, silicates and oxides of copper, and by that between any oxide and sulphide of copper, by which, in the one case, a matte was formed, and in the other metallic copper. The fuel used was anthracite; the pressure of blast to burn it produced a very high heat, but the furnace was so low that the reactions took place with great regularity, and but little difficulty was experienced in working it. The furnace had the apron in front somewhat longer, in order to make cleaner slags by allowing them to run into a reception basin. The operation was conducted at a high heat, and the furnaces were very low. The work was so rapidly done that little iron was reduced. If any but the purest products were used in this fusion, the black copper would have been very impure, as the heat was too high, and the furnace too low to get rid of impurities. The charge was calculated so as to give a scoria containing 40 per cent. of silica. The operation was conducted very rapidly; eight or nine tons of matte, equivalent to from fourteen to sixteen tons of charge, were fused in twenty-four hours.

The campaigns lasted only about twelve or fifteen days.

The charge was composed of:

Roasted mattes,	1000
Oxides of copper,	140
Rich roasted mattes,	60
	1200
Rich scorias,	80
Silicious scorias,	450
Debris of furnaces,	50
	1780

While the furnace was in operation between the castings the scorias ran continuously. They contained copper both as oxide and silicate, and, in addition, some small quantity of black copper. Those in which the copper was found as silicate or oxide contained over two per cent. however. These were sent to the matte fusion. Those which contained grains of black copper were passed in the same fusion. When the furnace was tapped the black copper sank to the bottom, the matte was on the top of that, and above was whatever scorias had run out of the tap-hole. As this was in contact with the matte it was quite rich and contained a considerable percentage of the matte in grains. It was cooled with water, broken up and put directly back into the top of the furnace. Below this was a small quantity of very rich and pure matte, which was taken off also in a single rosette, broken into small pieces, and sent to the roasting kilns. As there was but a small quantity of this it was scattered through the other mattes. The pieces were thin and easily recognized. These were all sufficiently roasted to be passed in the fusion for black copper in a single fire. This matte contained from 60 to 65 per cent. of copper.

The black copper contained from 90 to 92 per cent. of copper, considerable sulphur and iron. It varied in quality according to the ores from which it was made. Generally the quantity of impure ores added in the previous fusion was so very small that the impurities would all have been driven off, so that the black copper contained little or none of it. At first it was all cast as pigs in sand moulds, but toward the last it was found that it was possible to make better copper in the refining furnace by separating the black copper made from the various grades of ores. For this purpose the black copper was ladled into moulds, bearing a stamp which allowed of distinguishing the qualities, first, second, and third, according as it was made from pure, medium, or impure

ores. About 250 pounds of fuel were used for the ton of matte. This is a very small consumption of fuel for the quantity of roasted mattes and ores treated in the furnace. Below are given the expenses of a campaign of fifteen days, two furnaces working at a time:

Mattes treated,	180 tons
Rich carbonates and oxides,	13 "
Scorias,	85 "
Treated in the two furnaces,	278 "
Charcoal for two furnaces,	$10 00
Anthracite, 30 tons at $8.40,	252 00
Founder, 120 days at $1.50,	180 00
Common labor, 30 days at $1.25,	37 50
Repairs to two furnaces,	24 30
Cost of 278 tons,	$503 80

To get at the expenses for the ton of matte we must suppose the ores and the scorias converted into mattes. The ores are quite rich and contain at least 20 per cent. of copper. The scorias are also rich, and contain about 10 per cent. We suppose that the matte contains 50 per cent.:

13 tons of ores containing 20 per cent., or	5.2	tons of matte
85 tons of scorias containing 10 per cent., or	17	" "
Mattes treated,	180	
	202.2	

The 278 tons to be treated contain then 202.2 tons of matte. It is as though 202.2 tons of matte were treated, and as the price of the whole treatment is known, the price per ton of matte can be calculated:

Charcoal for lighting, 0.004 ton,	0.040
Anthracite, 0.148 ton,	1.244
Founder, 0.59 day,	0.826
Common labor, 0.15 day,	0.158
Repairs to furnaces,	0.120
Expense for 1 ton of matte,	2.388

To refer this expense to 1 ton of ore, it is only necessary to remember that 90 tons of ore gave 25 tons of matte. The 180

tons of mattes have been produced by $\frac{90 \times 180}{25} = 648$ tons of ore. To this 13 tons of ore have been added, or 661 tons in all. We have then:

Anthracite, 0.045 ton,	0.379
Charcoal, 0.001 ton,	0.01
Founder, 0.182 day,	0.254
Common labor, 0.009 day,	0.047
Repairs,	0.04
Cost per ton of ore,	0.730

The entire cost per ton of ore for the black copper is:

Wood for roasting,		0.0896
Charcoal, 0.0065 ton,		0.065
Anthracite, 0.222 ton,		1.8658
Roasting the ore, 0.15 Fusion for matte, 0.72, Roasting the matte, 0.125 2 fusions, 0.227	1.222 day,	1.5386
Repairs to furnaces,		0.1742
		3.7332

V. Fining and Refining.

The fining and refining of the black copper was done in a reverberatory furnace similar to the one used at Detroit, except that it had no opening in the roof:

Length of hearth,	10 feet 6 inches
Width of hearth,	8 feet
Length of fireplace,	4 feet 6 inches
Width of fireplace,	4 feet 3 inches

There were four of these furnaces at Canton, two of which were always at work. Later there were two at Point Shirley. The furnace has two doors, one at the end opposite the fireplace, which is the working door, and one in the center, which is the charging door. The hearth is inclined from the fireplace to the working door. The copper accumulates here; 8 tons of black copper are charged.

The fusion of the black copper took place at the end of fourteen to fifteen hours. In this first part of the operation the foreign metals contained in the copper, in melting slowly in an oxidizing

atmosphere, were oxidized. The temperature not being high some arsenic and antimony would have been driven off if there was any in the metal. The oxides of iron, copper, and sulphur were formed. When the fusion was complete the oxides of copper and iron formed scums on the surface, which were taken off, after having heated them almost to fusion, at which time the bricks of the furnace lost silica. In the furnace a reaction took place between the sulphide of iron and oxide of copper, and by this means the scums became poor when they were allowed to remain two or three hours on the melted copper at a high temperature. These scorias were taken off by means of an iron rod, which had a piece of wood at the end. They were put one side to be treated in the shaft furnace, as they always contained copper. The heat was kept up, but not so high as at first, in order to have an energetic action of oxidation, the object of which was to convert most of the foreign metals into oxides. This reaction was indirectly performed, for as the foreign metals were in very small quantity only, they were not easily reached by the action of the air. Copper oxide was formed with comparative rapidity, but was reduced by the foreign metals to metallic copper, and its oxygen transferred to them, so that they rose to the surface as oxides, and were at once acted on by the silica present, or, if volatile, they were carried off in the draft. The sulphur was concentrated in the copper until there were no longer any foreign oxides present. The copper oxides and sulphides reacted on each other, the copper was precipitated, and the gas set free with violent ebullition, which showed the end of the fining period. If the copper contained zinc, only a part of it would be driven off; a small quantity of zinc would injure its electrical conductivity, but would not otherwise hurt the copper. If the copper contained cobalt or nickel it first went into the scorias, and at the last part of the refining passed again into the copper.*

All the crasses and scorias which were removed during this part of the operation were very rich in copper, and were sent to the shaft furnaces for treatment. This action of oxidation was not stopped until the copper contained a considerable amount of oxide of copper. When a considerable quantity was dissolved there was no longer iron or sulphur. When the copper contained arsenic or

* Transactions American Institute Mining Engineers, vol. x, p. 49.

antimony lead was often added; this affected principally the antimony; one fourth of a pound of lead was added for every hundred pounds of copper. Sometimes as high as one per cent. was added. The lead combined with the antimony and sunk to the bottom. The copper contained less antimony, and did not contain any sensible amount of lead; that would injure it as much as the antimony All the lead could not, however, be removed. It should not generally be added in refining copper.* If there is much arsenic and antimony more can be sweated out than can be removed in any other way.† When the oxidation is complete the oxide of copper dissolved in the copper must be reduced. The fireman charges the fireplace, throws five or six shovelfuls of charcoal over the copper and puts in a stick of green wood. This produces an ebullition and mixes the lead and carbon with the copper. At the end of a quarter of an hour or twenty minutes assays are taken. When the oxide was reduced the copper was cast into the usual shape of the American ingot. Not more than five charges per week were made. The furnaces were stopped over Sunday, during which time they were banked. Sunday was always spent in making the necessary repairs to the hearth and sides. Not more than 250 charges per year could be made in the same furnace, as all of them had to be thoroughly repaired once a year.

There were always two furnaces which worked at a time. The work was so arranged that the charging took place at different hours, so that the same workmen might tend the two furnaces. At night two firemen only, one of whom was in charge of both furnaces, were required; during the day eleven men were required for the two furnaces. These were:

1 head refiner at,	$2 50
2 firemen,	1 50
8 men for charging and casting,	1 50

The cost of the treatment was:

Labor, 14 days at $1.50,	$21 00
Coal, 7 tons at $8.50,	59 50
Wood and charcoal, 0.185 tons,	6 85
Lead, 0.160 ton at $1.10,	17 60
Repairs,	2 80
Cost of the 2 furnaces,	$107 75

* Transactions American Institute Mining Engineers, vol. ix, p. 704.

† School of Mines Quarterly, vol. iv, p. 9.

The two furnaces refined 16 to 17 tons of black copper. The proportion of copper obtained depended on its purity. It was usually 81 to 82 per cent. The scorias were treated in the fusion for mattes.

The expenses for 1 ton of black copper were:

Labor, 0.875 day,	1.225
Coal, 0.437 ton,	3.816
Wood and charcoal, 0.53 ton,	0.424
Lead, 0.010,	1.1
Repairs,	0.175
Cost for 1 ton black copper,	6.740
Cost for 1 ton of ore,	0.955
We have found for the treatment of 1 ton of ore,	3.732
Cost of fining,	0.955
Special expenses for 1 ton of ore.	4.687

The ores yielded about 20 per cent. If the value of the copper was $500 per ton, a ton of ore yielded $\frac{500}{5}$ or $100.

Cost of treatment,	$4 687
General expenses,	4 00
	$8 687

At one time the expenses were found to be one-third labor, one-third fuel, and one-third all other expenses. Adding to this the price of the ore and we see a works advantageously situated to treat foreign ores when purchased at a low price.

This method, except for the fuel used, is essentially the same as that used at Ducktown.

THE ACTION OF HYDROFLUORIC ACID ON SILICA AND SILICATES (PRELIMINARY NOTE).

BY J. B. MACKINTOSH.

IT has been known for some time that some silicates, such as zircon, are not affected by hydrofluoric acid, and this fact has been utilized in order to purify crude zircons for analysis,* and to sepa-

* Linnemann, Chemical News, November 6th, 1885.

rate the various constituent minerals in rocks.* Messrs. Fouqué and Lévy state: "l'acide fluorhydrique attaque d'abord les matières amorphes, puis les feldspaths, puis le quartz cristallisé, et enfin les silicates ferrugineux et le fer oxydulé. . . . . Si, par exemple, on traite une ponce, mélange de matière amorphe et de minéraux cristallins, on attaque successivement la matière amorphe, les feldspaths, puis l'olivine, le fer oxydulé, l'amphibole, le pyroxène." My attention was first directed to this subject by finding that garnet was not affected in the slightest degree by the concentrated acid, and since then I have tried a larger number of silicates, about half of which are also unattacked. My results, however, differ somewhat from those of Fouqué and Lévy as quoted above. Generally speaking, those minerals which have a more condensed molecule are less easily attacked than those with a less condensed molecule, which fact is highly in favor of the "Natural System in Mineralogy," proposed by Dr. T. Sterry Hunt.† When Dr. Hunt, to whom I am indebted for suggestions and information, first learned that I was experimenting in this direction, he predicted correctly the action on some species, and suggested that I should make my experiments quantitative.

The results of these quantitative experiments are of very great interest. I will, however, only give here the difference between the two forms of silica, quartz and opal. When one gram of each, reduced to a uniform average size of $\frac{1}{100}$ cubic millimeter, was treated with an excess of dilute hydrofluoric acid (9 per cent.) in the cold for one hour, the quartz lost only 1.56 per cent., while the opal had lost 77.28 per cent. With crystals of quartz and pieces of opal of corresponding size, the difference is still more strongly marked. Even when treated for one hour in concentrated (54 per cent.) acid, quartz crystals suffered a diminution of weight of about 0.04 per cent. and 0.24 per cent, while in the dilute acid for the same length of time, opal lost 28.5 per cent. The differences in the amounts dissolved off different quartz crystals seems to have much more than merely chemical significance, and to be connected with those molecular dissimilarities which cause the differences between right and left handed quartz, the occurrence of plagihedral faces, and of those rhombohedral faces which differ in angle from the primitive form.

* Minéralogie Micrographique, F. Fouqué et Michel Lévy. Paris, 1879, p. 116.

† See Review on following page.

BOOK REVIEWS.

A NATURAL SYSTEM IN MINERALOGY; WITH A CLASSIFICATION OF NATIVE SILICATES. By Thomas Sterry Hunt, M.A., LL.D. (Cantab.) From the Transactions of the Royal Society of Canada. Vol. III, Sec. III. 1885. 69 pp., 4to.

"If, however, we regard as mistaken those who in their search after a natural system in mineralogy have rejected the aid of chemistry, it must be said on the other hand, that the chemical mineralogists, who, disregarding the relations of density and hardness or relegating them to a secondary rank, build systems on the results of chemical analysis, are false to chemical science itself. There exist, in fact, inherent and necessary relations between the physical characters and the chemical constitution of inorganic bodies, which serve to unite and reconcile the natural-historical and the chemical methods in mineralogy."

The occurrence in nature of isomeric compounds, such as quartz, tridymite and opal, rutile, brookite and anatase, calcite and aragonite, diamond and graphite. and many other instances, in which chemical analysis can detect but little difference, while the physical properties are very distinct, renders a mineralogical classification, based on chemical composition alone, insufficient to properly explain the occurrence of these species. Dr. Hunt in the present paper, which is but a portion of a more extensive work now in preparation, aims to place the science of mineralogy on a more secure foundation by giving a reason for these differences. The only reasons which it is possible to give in these cases are that the number of atoms in the molecule is greater in one case than the other, or that the number remaining the same the method of arrangement varies, or that both variations take place simultaneously. In either way of considering the case we have in the specific gravities of allotropic forms a measure of the relative number of atoms present in equal volumes, or in other words of the amount of condensation of the molecule. Thus in the case of the three forms of silica, quartz, tridymite and opal, with the respective specific gravities of 2.67, 2.32, and 2 (?), the number of atoms in the molecule, or in the aggregate of molecules, varies directly as these numbers, and the specific volumes inversely. With this varying condensation, the physical properties of the mineral, and to a smaller extent, perhaps, the chemical properties, vary correspondingly. The hardness, index of refraction and resistance to chemical action, are all greater with the greater condensation of the molecule. In the case of quartz and opal the latter is readily dissolved by caustic alkalies and by hydrofluoric acid,* while the former is hardly affected.

Calcite and aragonite offer a very interesting instance of these differences. The comparison is best seen in a tabular arrangement:

	Specific Gravity.	Hardness.	Index of Refraction.	Chemical Action.
Calcite.....	2.72	3	1.664 Ordinary. 1.489 Extraordinary.	Dilute HCl dissolves Calcite more readily than Aragonite.
Aragonite..	2.94	3½-4	1.686 mean	

* See previous page.

With diamond and graphite the chemical difference is more strongly marked, as is shown by the formation of graphitic acid.

In all these cases the differences observed are analogous to the differences between various members of the olefine series of hydrocarbons (C_nH_{2n}). In the case of some silicates it may be considered that they are members of series analogous to other carbon series ($C_nH_{2n\pm x}$) in which there is a first term different from the regular increment. Other silicates, again, are analogous to the C_nH_{2n} series, as are the minerals previously instanced.

The recent researches of Wolcott Gibbs have shown that in the domain of inorganic chemistry we may have definite compounds, as complex as any of those with which the organic chemist is familiar; in the vanadates, alone, we may combine with nR_2O, amounts of vanadic acid, varying from $(2n+2)V_2O_5$ to $(2n-6)V_2O_5$, in which the value of n may vary from 1 to 7. To illustrate the extraordinary complexity of some of these inorganic compounds Gibbs has described a crystalline phospho-vanadio-vanadico-tungstate of barium, the formula of which is $60WO_3, 3P_2O_5, V_2O_5, VO_2, 18BaO + 150H_2O$, with a molecular weight of twenty thousand and sixty-six (20,066). It is to be remembered that this number, large as it is, is only the smallest possible number for the molecular weight of this compound, and that the true molecular weight may be some multiple of this figure. The small amounts of phosphorus and vanadium (in each case less than one per cent.) forming essential parts of the compound, will "by an analogy not too remote," help us "to understand the part played in many natural silicates by small portions of fluorine, of chlorine, of sulphur, of sulphates and carbonates, and even of phosphates, arsenates and vanadates, which have hitherto been in many cases regarded as accidental impurities."

The classification of mineral species proposed by Dr. Hunt is as follows:

Classes.	Orders and Sub-orders.
I.	1. METALLATES: *a.* Metallometallates; *b.* Spathometallates.
II.	2. OXYDATES. 3. SILICATES: *a.* Protosilicates; *b.* Protopersilicates; *c.* Persilicates. 4. TITANATES. 5. NIOBATES. 6. TANTALATES. 7. TUNGSTATES. 8. MOLYBDATES. 9. CHROMATES. 10. VANADATES. 11. ANTIMONATES. 12. ARSENATES. 13. PHOSPHATES. 14. NITRATES. 15. SULPHATES. 16. BORATES. 17. CARBONATES. 18. OXALATES.
III.	19. HALOIDATES: *a.* Fluorides; *b.* Chlorides; *c.* Bromides; *d.* Iodides.
IV.	20. PYROCAUSTATES: *a.* Carbates; *b.* Carbhydrates.

The order SILICATES, with its sub-orders, is divided into the following tribes:

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Order.	Sub-order.	Tribe.	Tribe Volume.	Example.
Silicate.	a. Protosilicates.	1. Pectolitoid (Hydro-proto-spathoid)..	7.0–5.3	Calamine.
		2. Protospathoid.....	6.7–6.0	Willemite.
		3. Prot-adamantoid.....	6.0–4.6	Chrysolite, Pyroxene.
		4. Proto-phylloid.....	...	Talc.
		5. Ophitoid (Protocolloid).....	7.3–5.5	Serpentine, Chrysocolla.
	b. Protopersilicates.	6. Zeolitoid (Hydro-proto-per-spathoid.....	7.2–6.3	Zeolites.
		7. Proto-per spathoid.....	8.6–6.1	Felspars.
		8. Proto-per-adamantoid.....	5.8–4.7	Garnet, Spodumene.
		9. Proto per-phylloid.....	6.2–5.1	Micas.
		10. Pinitoid (Proto-per-colloid).....	...	Pinite, Obsidian.
	c. Persilicates.	11. Hydro-per-spathoid.....	...	Westanite.
		12. Per-spathoid.....	7.7–7.2	Bismutoferrite, Eulylite.
		13. Per adamantoid.....	5.7–4.4	Topaz, Cyanite, Zircon.
		14. Per-phylloid.....	6.1–5.1	Pyrophyllite.
		15. Argilloid (Per-colloid).....	6.5–5.4	Kaolin.

†

CLASS-BOOK OF GEOLOGY. By Archibald Geikie, LL.D., F.R.S. MACMILLAN & CO. 1886. 516 pp.

The broad discussion of the facts of any science is usually left to the lecture room, and the text-books concern themselves chiefly with the details. For this reason text-books are but dry reading, and unless supplemented by careful instruction, are more apt to destroy than to increase the interest of the learner.

Professor Geikie's Class-book of Geology avoids detail where possible, and thus succeeds in presenting a series of clear, interesting views of the various branches of the subject. The book is divided into four parts, and perhaps the most interesting of these is Part I. "The Materials for the History of the Earth." Here he discusses with great clearness the action of the atmosphere and of running water, the records left by Seas, Lakes, Hot Springs, Volcanoes, Plants, and Animals, and how these records are read. Part II. describes briefly the chief minerals and rocks of the earth's crust. Part III. discusses the structure of the rocks, and tells how fossils have been entombed and preserved, and how they are used in geological study. Part IV. describes the different geological periods.

The work is clearly printed, on heavy paper, and illustrated by excellent wood-engravings, many of which are from landscape sketches by the author. *

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